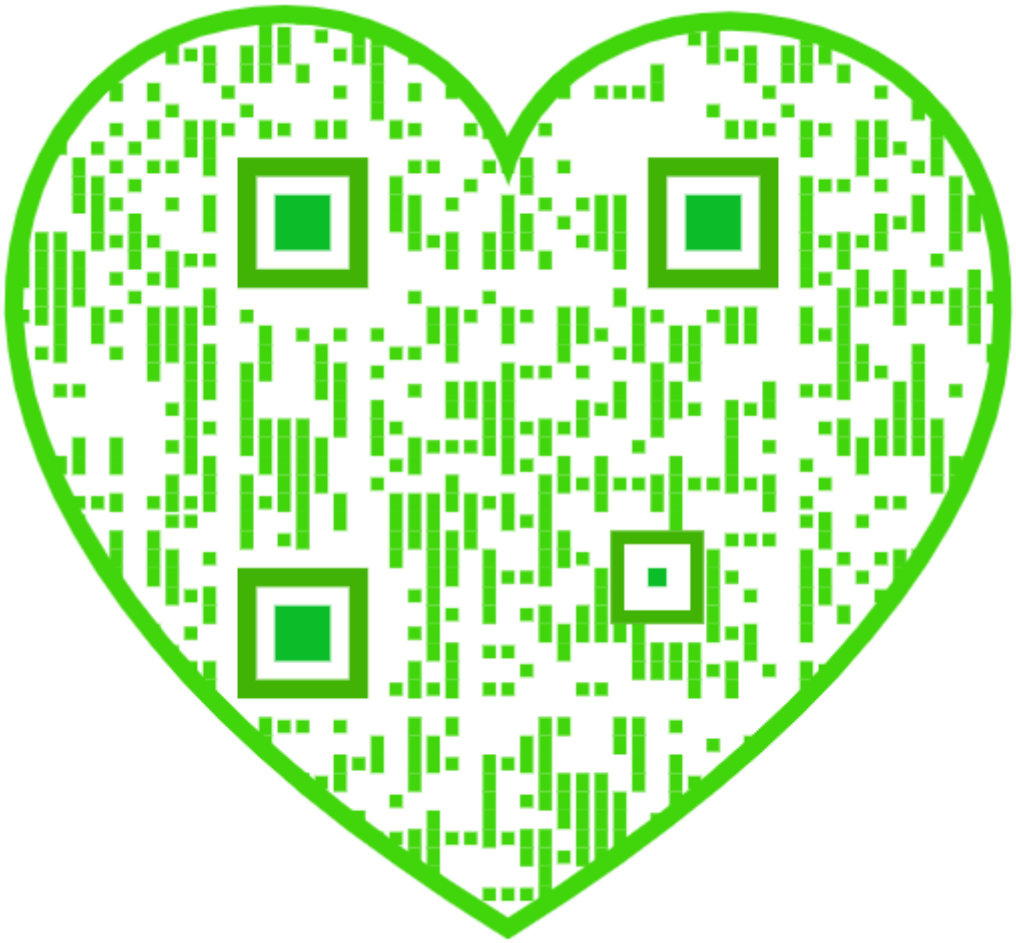
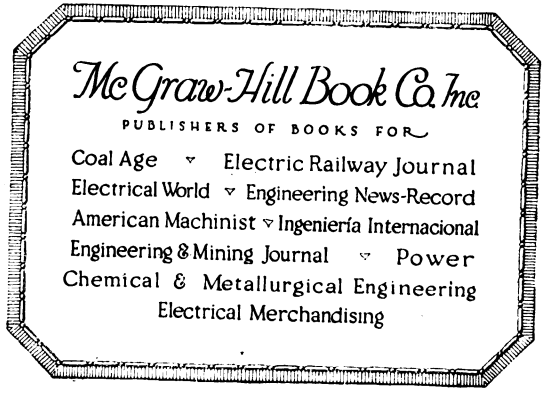


HANDBOOK OF ORE DRESSING

EQUIPMENT AND PRACTICE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HANDBOOK OF ORE DRESSING

EQUIPMENT AND PRACTICE

BY

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PREFACE

In the following pages an attempt has been made to cover, in condensed form, the various stages in the mechanical handling and preparation of an ore for metallurgical treatment. Amalgamation, concentration, flotation, chemical solution, and smelting—by which either a metal or a mineral is extracted from an ore or concentrate—are considered as essentially metallurgical processes, deserving of specialized treatment, and outside the scope of the book. A line of demarcation between ore dressing and metallurgical extraction and recovery is proposed which, it is hoped, may be found acceptable, as aiding in the more precise interpretation of terms, and the better classification of data.

Information and illustrations have been obtained from a variety of sources, and a bibliography is appended. Appreciative acknowledgment is made to those mentioned, as well as to the publishers, and apologies are tendered to any whose names have been inadvertently omitted.

The aim has been to supply a handy and practical *vade-mecum* for millmen and engineers. The author shares that fallibility which almost invariably characterizes an effort of this kind. He will welcome and appreciate constructive criticism and comment, and the disclosure of any errors that may be noticed.

A. W. A.

NEW YORK,
Dec. 1919.

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HANDBOOK OF
ORE DRESSING

EQUIPMENT AND PRACTICE

SECTION I

INTRODUCTION

The extended ramifications of metallurgical practice and the increasing importance of specialized methods of extraction and recovery of metals and concentrates suggest the necessity for an acceptable definition of the phrase "ore dressing." The fundamental idea underlying the verb "to dress," is "to put in good order" or "to make ready." A second sense, perhaps of wider and older application in metallurgy, is "to cleanse from impurities," and a typical use of the term in this connection is seen in the phrase "the dressing of tin." Tin in this instance refers to the more important tin mineral, cassiterite, known as black tin. The dressing of graphite covers the operations involved in the preparation of the ore and the removal of the valueless material. The graphite is cleansed from associated impurities; hence the phrase, which has a definite meaning. "The dressing of minerals" has been defined[1] as including "all operations that intervene between the extraction of a mineral from its natural deposit and its production in a condition ready for sale or for further use in the arts or manufactures." The treatise in which this definition is found covers crushing and allied operations, as well as concentration. Amalgamation is considered a metallurgical operation, and so outside the scope

of the book. In commenting on this it may be mentioned that various processes have become standard practice since the time when simple concentration of mineral was performed only by gravity methods; so much so that it is difficult to maintain the distinction, or to suggest, for instance, that flotation is less of a metallurgical operation than amalgamation. It will be noted that there is no mention of ore. Without a radical modification in the accepted interpretation of the verb "to dress," the impracticability of a combination of the two words may have been realized. On the other hand, the phrase "ore dressing" was apparently adopted by a number of writers to cover all the different processes involved in the dressing of minerals and metals.

A recognized authority defines[2] "ore dressing" as "the preparation of the ore for the smelter by mechanical means whereby the valuable minerals are concentrated into a smaller bulk and weight by the separation of some of the waste, or whereby two valuable minerals are separated from each other." A detailed criticism of this definition might result in an involved discussion as to acceptable interpretations of "ore" and "waste"; and the first difficulty in dealing with the phrase is encountered. To cleanse a mineral like graphite from impurities means the removal of non-graphitic material; and this is insured by a series of graphite-dressing operations. To cleanse an ore should necessitate nothing more than the removal of waste, as distinct from ore; and this is usually achieved by picking and washing, accompanied sometimes by primary crushing. Hence the need of explaining that, whereas "graphite dressing" means the dressing of graphite, "ore dressing" has been adopted to mean much more than the dressing of ore.

A second definition then followed[3] in which ore dressing was defined as "the process of mechanically separating and saving valuable minerals from the valueless material of an ore whereby its valuable minerals are concentrated into smaller bulk and weight by discarding some of the waste, or the process of mechanically separating two or more minerals, which combined have little value, into two or more products, each of increased value."

The various processes involved in the extraction and recovery of the valuable portions of an ore have specific designations. The problem is to know where to draw the line as to which of them is to be included in ore dressing. The treatises in which these two latter-mentioned definitions are found make especial mention of amalgamation and concentration as phases of ore dressing; but the definition would also include other specialized recovery processes. The same authority defines "coal dressing" as "the removal from coal of the impurities which it always contains." This is a literal adaptation of one of the dictionary definitions of the term. If the interpretation "to cleanse from impurities" is to be taken as definitive of the verb "to dress," then the term "ore dressing" cannot logically be used to represent operations such as graphite dressing, gold dressing or coal dressing.

An alternative definition of ore dressing is inferred where the term is used[4] to refer to "the elimination of the waste portions of metallic ores by mechanical means and without the intervention of the chemical processes of solution, combination or precipitation, or the employment of high temperatures except for roasting or drying."

Sufficient evidence has been brought forward to show that there is a lack of professional agreement as to the meaning of the phrase. This leads to confusion, and prohibits proper classification alike of both ideas and data. The term "ore dressing" has been retained in connection with the reduction and treatment of ores, whereas the processes concerned have undergone extensive modifications and expansions. New methods and inventions have been adopted, more particularly in connection with the extraction and recovery of minerals and metals, for which new and appropriate designatory terms have become necessary. The fact that a number of these processes are still included under the many existing definitions of "ore dressing" is the cause of much confusion; and a reconsideration of the subject is pertinent. One noticeable result of the uncertainty as to the scope of the term is seen in the fact that it is now seldom used by practical

engineers when writing on technical subjects. One of the most important papers, by A. P. Watt, published during 1917 was entitled "Concentration Practice," whereas it referred largely to ore-dressing operations, as distinct from any specialized method of mineral separation or extraction. The result of the uncertainty that exists is that much valuable data are unsuitably indexed and cannot be found when needed. According to the "Engineering Index" authorities (1918), smelting is an ore-dressing operation.

To define the scope of ore dressing logically and authoritatively in harmony with existing definitions is an impossibility. The published interpretations are so varied, and encroach, in many instances, on recovery processes, that a new departure is permissible. A permanent and radical alteration in interpretation is preferable to an attempt to expand periodically the definition, so as to embrace or exclude new treatment processes; and it is suggested that the subject may be considered as "that branch of metallurgy covering the reduction or other mechanical handling of the ore whereby one or more products are obtained in a condition to be treated for the isolation of their valuable contents by amalgamation, concentration, wet-chemical, smelting, or other process of recovery."

To summarize the contentions which have been advanced:

1. There are two interpretations of the verb "to dress," one of which means "to cleanse from impurities." To remove waste from coal is termed "coal dressing;" and to remove gangue from graphite is termed "graphite dressing." The general adoption of the term in this sense would be too cumbersome, as individual designations would be needed in connection with the dressing of each metal or mineral. Interpreted in this way, the use of the word in conjunction with the word "ore," as "ore dressing," has led to confusion and inconsistency.

2. The alternative meaning of the verb "to dress" is "to prepare" or "to arrange." In this sense it is most applicable in the present instance; and, in this treatise, ore dressing is a phrase used only to refer to those preliminary mechanical or semi-mechanical operations that precede or operate concurrently with a specialized method of extraction or recovery.

Overlapping of ore dressing with metallurgical extraction is inevitable, but as a general rule a fairly distinct line of demarcation may be traced Examples of the simultaneous operation of ore dressing and extraction are seen when an ore is crushed in cyanide solution, or amalgamated in a grinding pan. Roasting occurs as an ore-dressing operation in the preparation of a sulphotelluride ore for cyanide treatment, and as a method of extracting the metal in the volatilization process.

The treatment of the subject of ore dressing may be conveniently considered under the following sub-sections:

(i) Ore conveyance and storage, elevation, feeding, and sampling.
(ii) Ore tonnage and moisture estimation.
(iii) Theory of crushing.
(iv) Roasting.
(v) Screening.
(vi) Coarse or medium crushing in jaw, gyratory, disc or other machines.
(vii) Stamp milling.
(viii) Ball milling.
(ix) Grinding in Chilean, Huntington, and similar types of mills.
(x) Roll crushing.
(xi) Regrinding or sliming in pans and tube-mills.
(xii) Conveyance and elevation of ore pulp
(xiii) Classification of ore pulp.
(xiv) Thickening, settling, and dewatering of ore pulp.

SECTION II

ORE CONVEYANCE AND STORAGE, ELEVATION, FEEDING AND SAMPLING

Ore is generally **Transported to the Rock Crusher** in steel cars of various sizes and shapes, usually mounted on four-wheeled trucks. **Rocker Side-Dump Cars** are largely used for the handling of ore, and may be tipped at either side. Table I gives data with

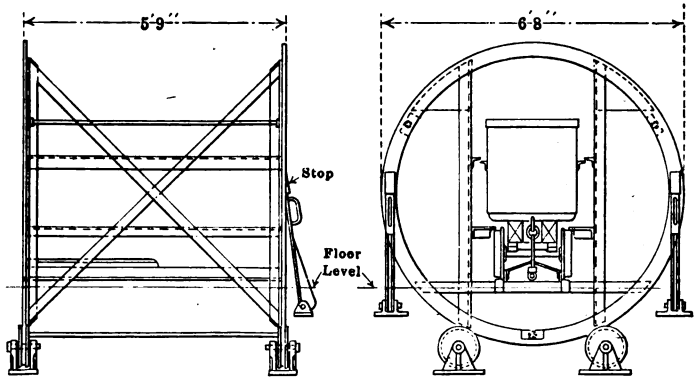


FIG. 1.—Revolving tippler.

reference to **Round-Bottom Rotary Dump Cars. Gable Cars** are useful in cases where it is preferable to tip on both sides of the track simultaneously. **Railroad Dump Cars** are usually adopted for ore conveyance to large mills.

A **Revolving Tippler** for ore cars is shown in Fig. 1. The apparatus consists of a framework, carrying rails connecting two points in the track, and mounted by tires and rollers in the manner shown. The empty car is pushed out by the incoming full car, and the framework is balanced to tip automatically, under control of a hand brake.

TABLE I.—ROUND-BOTTOM ROTARY DUMP CARS FOR 18-IN. TRACK[5]

Capacity, cu. ft.	Wheel		Wheel base, in.	Dimensions of box			Thickness of steel, in.	Shipping weight, lb.
	Diam., in.	Tread, in.		Length, in.	Width, in.	Depth, in.		
9	10	2¾	14	36	26	18	⅛	650
12	10	2¾	16	48	26	18	3/16	725
14	10	2¾	16	48	26	21	3/16	750
14	12	3	16	48	26	21	3/16	875
16	10	2¾	16	48	26	24	3/16	770
16	12	3	16	48	26	24	3/16	900
18	10	2¾	16	48	28	24	3/16	790
18	12	3	16	48	28	24	3/16	920
20	12	3	16	48	30	24	3/16	930
24	12	3	16	48	36	24	3/16	970
27	12	3	18	54	36	24	3/16	1000
30	12	3	18	60	36	24	3/16	1040
32	12	3	18	60	36	26	3/16	1075

Ore Bins are designed either with flat or sloping bottoms. Flat-bottomed bins are favored by those who maintain that the design allows for an accumulation of "dead" ore for use in the event of an emergency. Sloping-bottomed bins are generally to be preferred and should be designed of a capacity to allow for any reasonable delay due to temporary stoppage of mine operations, the necessity for making rock-breaker repairs, or other emergencies. Large ore bins that will empty without the aid of hand labor are almost invariably a satisfactory investment. The slope of the bin bottom should be about 45 degrees.

Steel Ore Bins are generally made in one of two types, viz.: (*a*) having steel framework supporting comparatively flat sides and bottom plates, and (*b*) cylindrical bins, which are of simpler and cheaper construction. Height, in the latter type, is out of proportion to capacity, but this objection is counterbalanced by the fact that, if properly designed and stiffened, a series of these cylindrical bins will serve as adequate support for a light railroad.

The **Erection Expense and Cost of Materials** of sloping-bottom ore bins are considerably more than with flat-bottomed bins.

There is also a higher cost due to increased wear and the necessity for the frequent renewal of steel plate or other lining.

The **Approximate Specific Gravities and Weights Per Cubic Foot** of various rocks and other materials handled in ore-dressing operations are given in Table II.

TABLE II.—APPROXIMATE SPECIFIC GRAVITY AND WEIGHT OF VARIOUS SUBSTANCES[6]

	Sp. gr., av.	Av. wt. per cu. ft.		Sp. gr., av.	Av. wt. per cu. ft.
Earth, etc., excavated			**Minerals**		
Clay, dry	1.0	63	Asbestos	2.45	153
Clay, damp, plastic	1.76	110	Barytes	4.5	281
Clay and gravel, dry	1.6	100	Basalt	3.0	184
Earth, dry, loose	1.2	76	Bauxite	2.55	159
Earth, dry, packed	1.5	95	Borax	1.75	109
Earth, moist, loose	1.3	78	Chalk	2.2	137
Earth, moist, packed	1.6	96	Clay, marl	2.2	137
Earth, mud, flowing	1.7	108	Dolomite	2.9	181
Earth, mud, packed	1.8	115	Feldspar, orthoclase	2.55	159
Riprap, limestone	1.35	82	Gneiss, serpentine	2.55	159
Riprap, sandstone	1.4	90	Granite, syenite	2.8	175
Riprap, shale	1.7	105	Greenstone, trap	3.0	187
Sand, gravel, dry, loose	1.55	100	Gypsum, alabaster	2.55	159
Sand, gravel, dry, packed	1.75	110	Hornblende	3.0	187
Sand, gravel, wet	1.9	119	Limestone	2.66	166
			Marble	2.65	165
Excavations in water			Magnesite	3.0	187
Sand or gravel	0.96	60	Phosphate rock, apatite	3.2	200
Sand or gravel and clay	1.00	65	Porphyry	2.75	172
Clay	1.28	80	Pumice	0.62	40
River mud	1.44	90	Quartz, flint	2.65	165
Soil	1.12	70	Sandstone, bluestone	2.35	147
Stone riprap	1.00	65	Shale, slate	2.75	172
			Soapstone, talc	2.7	169
Coal, coke, etc.			**Quarried, piled**		
Anthracite	0.84	53	Basalt, granite, gneiss	1.5	96
Bituminous, lignite	0.75	47	Limestone, marble, quartz	1.5	96
Peat, turf	0.37	23	Sandstone	1.3	82
Charcoal	0.20	12	Shale	1.5	92
Coke	0.44	28	Greenstone, hornblende	1.7	107

Screw Spiral Conveyors may be used for the transference of ground ore from one part of the mill to another, or in connection with the feeding of this material to fine-grinding or other machinery. The wear is usually considerable, and the cost of maintenance higher than with other types. It is a suitable system

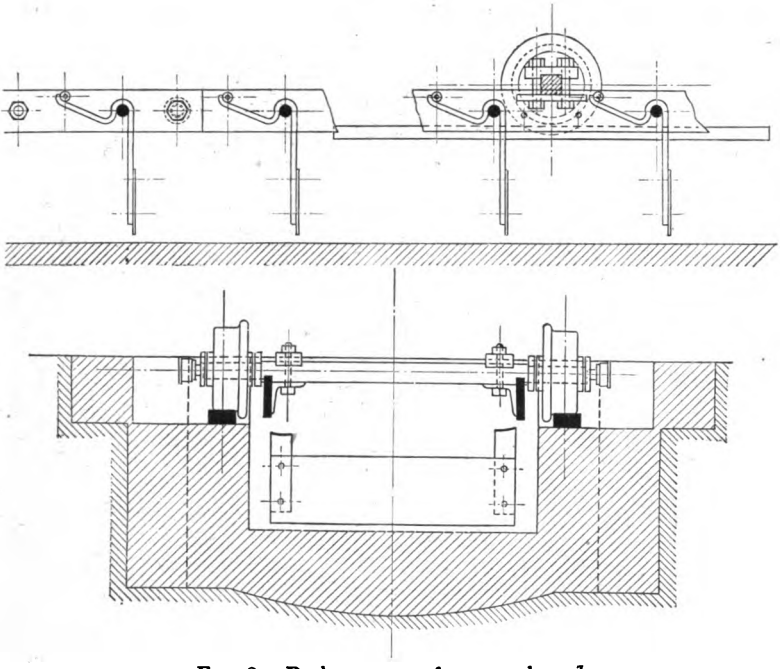


FIG. 2.—Push conveyor for ground ore.[7]

for the conveyance of ore after roasting. A 6-in. diameter spiral at 120 r.p.m. will convey about 120 cu. ft. of material per hour. A 9-in. spiral at 100 r.p.m. will convey about 400 cu. ft. in the same time.

Details of a **Push Conveyor** are given in Fig. 2. The fine ore is fed by this contrivance to a Merton-furnace feed channel. The stroke, effected by means of a crank at the head of the conveyor and operated and controlled by suitable gearing, is 18 in. and

the frequency 20 per minute. The wearing faces consist of 13-gauge renewable steel plate. Several other classes of push conveyors are available, but have no wide application.

Belt Conveyor Data will be found in Table III. The same authority (C. Kemble Baldwin) gives the following formulæ

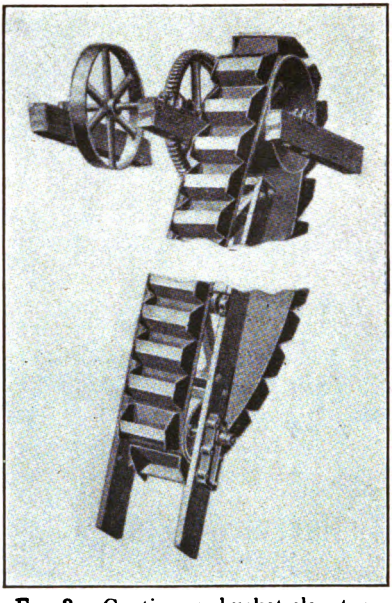


Fig. 3.—Continuous bucket elevator.

for power consumption, where C equals the power constant; T the load in tons of 2000 lb. per hour; H the vertical height the material is lifted in feet, then:

for level conveyors h.p. $= C\ T\ L\ /1000$

for inclined conveyors h.p. $= (C\ T\ L/1000 + (T\ H/1000)$

Conveyor Belt Wear may be reduced in the case where the material is fairly dry and contains an evenly distributed quantity of fines by placing a small grizzly in the feed chute to form a bed on the belt to receive the coarser portions of the ore.

A Continuous Bucket Elevator, suitable for raising broken ore, is shown in Fig. 3. Table IV gives sizes and capacities.

TABLE III.—DATA ON BELT CONVEYORS[6]

Size		Speed	Capacity		Power			Belts		Idlers	
Width of belt in inches	Max. size of pieces in inches	Max. advisable speed in feet per minute	Capacity in cu. ft. per hr. at belt speed of 100 ft. per minute	Capacity in tons per hr. at belt speed of 100 ft. per min. cap. of material weighing 100 lb. per cu. ft.	Constant C for material weighing, per cu. ft.		H.p. required for each movable or fixed tripper	Min. ply	Max. ply	Spacing of troughing idlers in ft. for material weighing, per cu. ft.	
					50 lb.	100 lb.				50 lb.	100 lb.
12	2	300	460	23	0.177	0.127	½	3	4	5	4½
14	2½	300	630	31	0.175	0.124	½	3	4	5	4½
16	3	300	820	41	0.172	0.121	¾	4	5	5	4½
18	4	350	1040	52	0.160	0.116	1	4	5	4½	4½
20	5	350	1280	64	0.189	0.133	1¼	4	6	4½	4½
24	8	400	1850	93	0.181	0.130	1¾	4	7	4½	4
30	12	450	2900	145	0.161	0.119	2½	5	7	4½	4
36	18	500	4200	210	0.150	0.116	3¼	5	8	4	3½
42	20	550	5600	280	0.147	0.112	4¼	7	10	4	3
48	24	600	7400	370	0.138	0.106	5	7	11	3½	3

Intermittent Bucket belt-and-bucket elevators are in common use. Table V gives details of their operation, either with broken ore or mill pulp.

TABLE IV.—CONTINUOUS-BUCKET ELEVATOR DATA[8]

Maximum capacity of elevator in tons per hour	Length between centers of head and foot shaft, ft.	Size and gauge of buckets, in.	Width of belt, in.	Total weight, lb.	R.p.m. pinion shaft	R.p.m. head shaft
30	30	9× 9 No. 16	10	3400	171	32
50	30	13×10 No. 14	14	4900	197	32
80	30	15×11 No. 14	16	5900	154	27
120	30	18×13 No. 12	20	6300	131	23
200	30	24×14 No. 12	26	8100	135	21
325	30	30×17 No. 10	32	8500	123	19
450	30	36×18½ No. 7	38	9500	109	19
600	30	42×19 3/16 in.	44	10500	112	19
700	30	48×20 3/16 in.	50	11500	112	19

TABLE V.—STANDARD VERTICAL MILL ELEVATORS[11]

Size of Buckets, Bucket Belt, Head, Boot, Driving Pulleys and Speed

Buckets				Bucket belt		Head pulley		Boot pulley		T. and L. driving pulleys, and r.p.m.								
										Elevators up to 35-ft. centers						Elevators 35- to 60-ft. centers		
										Single head			Geared head			Geared head		
Length, in.	Width, in.	Depth, in.	Style	Width, in.	Ply	Dia., in.	Face, in.	Dia., in.	Face, in.	Dia., in.	Face, in.	R.p.m.	Dia., in.	Face, in.	R.p.m.	Dia., in.	Face, in.	R.p.m.
6	4	4¼	A	8	5	24	9	20	9	28	4½	48	18	4½	144	18	4½	144
8	5	5½	A	10	5	28	11	24	11	32	4½	41	22	4½	123	22	4½	123
10	6	6¼	A	12	6	32	13	28	13	36	6½	36	26	4½	108	26	6½	108
12	7	7¼	A	14	6	36	15	32	15	40	10½	32	30	4½	96	30	6½	96
14	7	7¼	A	16	8	40	17	36	17	44	12½	29	30	6½	116	30	8½	116
16	7	7¼	A	18	8	44	19	40	19	48	12½	26	34	6½	104	34	8½	104
18	8	7⅝	AA	20	8	48	21	44	21	52	14½	24	34	6½	120	34	8½	120
20	8	7⅝	AA	22	8	52	23	48	23	56	14½	22	36	6½	110	36	8½	113

STANDARD VERTICAL MILL ELEVATORS (*continued*)

Capacities and Horse Power

Capacities: 20 cu. ft. of ore = 1 ton								H.p. required, buckets full	
Size of bucket, in.	Cap. of bucket, cu. in.	Dry material, buckets ⅓ full		Dry material, buckets ½ full		Tons of mineral per hour, buckets ½ full			
		Cu. ft. per hour	Tons per hour	Cu. ft. per hour	Tons per hour	Water 3 mineral 1	Water 5, mineral 1	Up to 35-ft. centers	35-ft. to 60-ft. centers
6	55	168	6.4	192	9.6	2.4	1.6	1.0	2.0
8	115	268	13.4	402	20.1	5.0	3.3	2.0	3.5
10	204	474	23.7	711	35.5	8.9	5.9	3.5	6.5
12	332	770	38.5	1155	57.7	14.4	9.6	6.0	10.0
14	391	906	45.3	1359	67.9	16.9	11.3	7.5	12.5
16	467	1082	54.1	1623	81.1	20.2	13.5	8.5	15.0
18	500	1158	57.9	1737	86.8	21.7	14.4	9.5	16.0
20	560	1292	64.6	1938	91.9	24.3	15.3	10.0	17.0

Buckets are spaced 18 in. apart.
Speed of belt 300 ft. per minute.
Horse powers indicated include 5 per cent. for friction.

Speed, Capacity, and Size of Bucket Elevators, according to Robins practice, may be found from the bucket-elevator chart given in Fig. 5. To find the h.p. required to operate a vertical-bucket elevator, multiply tons of ore and water per hour by lift in feet and divide by 500. The figure obtained includes an allowance of 100 per cent. of useful work for friction. To find the proper size of a bucket elevator for a given tonnage or volume of material per hour and given weight per cubic foot, commence at left of chart with tons per hour and follow across to diagonal line showing weight per cubic foot, thence down to diagonal line representing the desired speed of elevator, thence across to the right until the diagonal showing the desired spacing of buckets is reached. At the bottom, directly below this intersection, is found the required capacity of bucket in cubic inches. If the hourly capacity is given in cubic feet, commence at the top of the chart and follow down until the desired speed line is found, thence across to spacing as before. The heavy line is an example showing that 65 tons per hour of a material weighing 50 lb. to the cubic foot, with a belt speed of 175 ft. per minute,

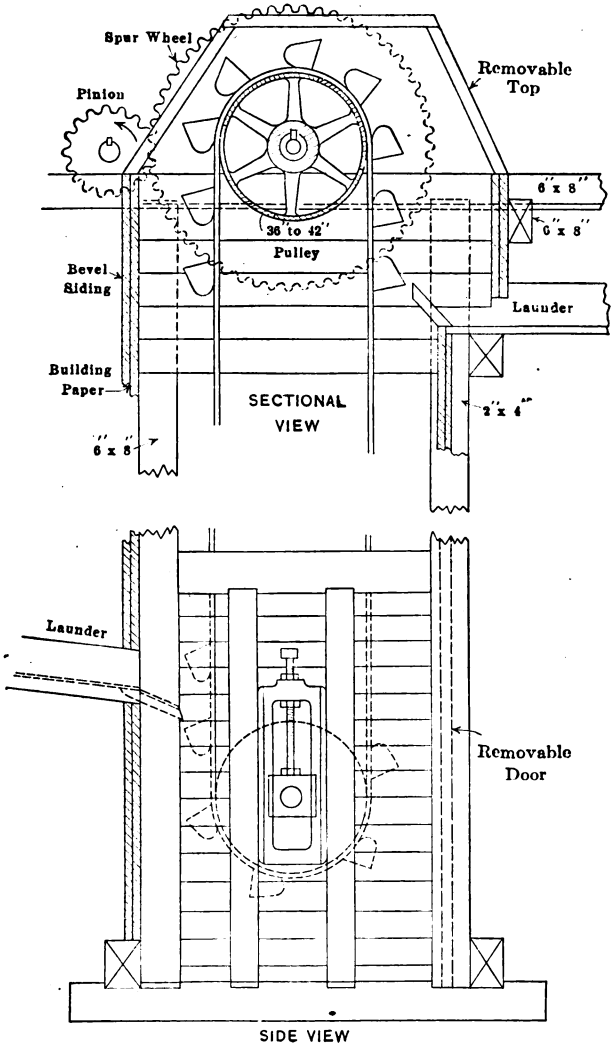


FIG. 4.—Belt-and-bucket elevator.

and 21-in. bucket spacing, will require a working capacity of 740 cubic inches.

Belt-Elevator Speeds for pulleys of various diameters may be calculated from the chart given in Fig. 6. **Width and Ply of**

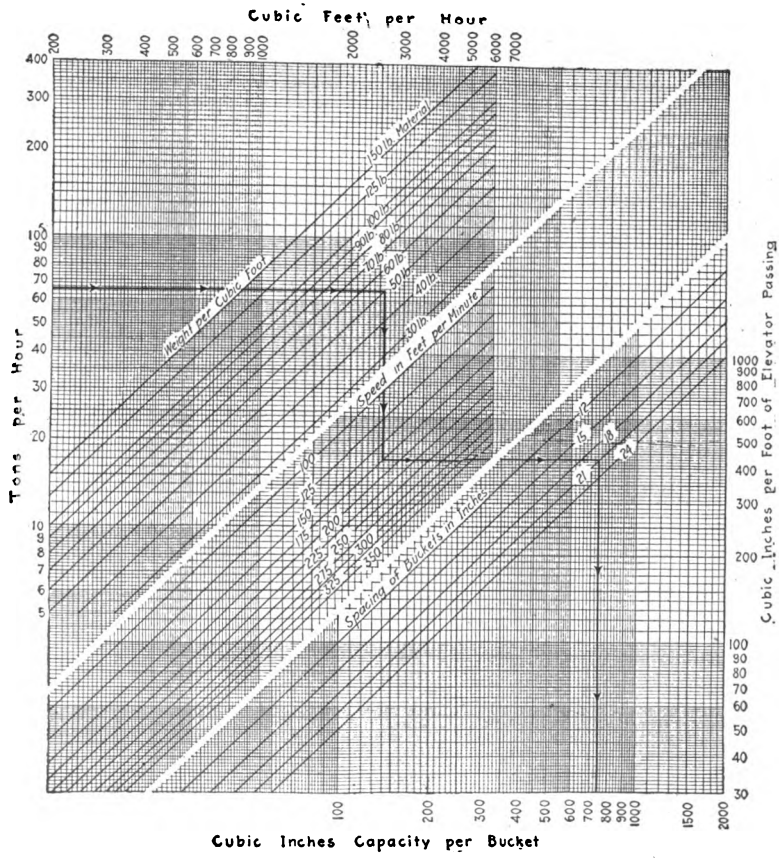

Cubic Feet per Hour
Tons per Hour
Cubic Inches Capacity per Bucket
Cubic Inches per Foot of Elevator Passing
Weight per Cubic Foot
Speed in Feet per Minute
Spacing of Buckets in Inches

FIG. 5.—Bucket elevator chart.[9]

Belting used in connection with belt-and-bucket elevator operation are given in Table VI.

TABLE VI.—WIDTH AND PLY OF BELTING USED IN BELT ELEVATORS [10]

Height c.–c., ft.	Min. diam., in.	Recommended diam., in.	Belt, no. plies	Height c.–c., ft.	Min. diam., in.	Recommended diam., in.	Belt, no. plies
20	24	30	5	80	60	84	12
30	30	36	6	90	66	96	12
40	36	48	7	100	72	108	14
50	42	54	8	120	84	120	16
60	48	60	10	140	96	132	18
70	54	72	10	160	108	144	20

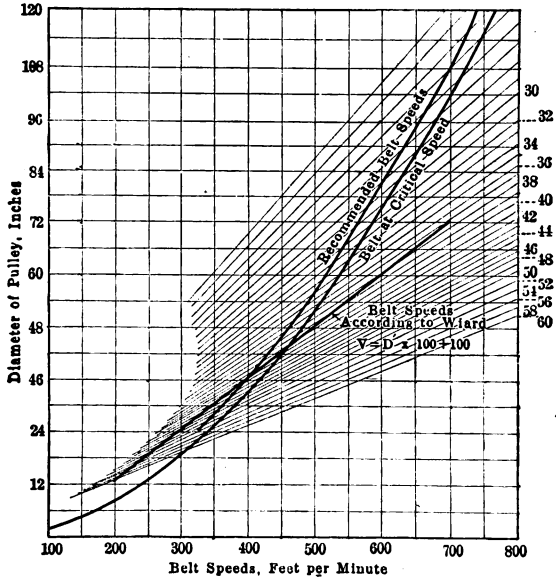


FIG. 6.—Belt elevator speeds for various pulley diameters.[10]

Power for Belt-and-Bucket Elevators is usually applied through gearing, as shown in Fig. 7. **Take-up Boxes** (Fig. 8) are used to allow for the stretching of the belt, and obviate the necessity for frequent shut-downs on this account. The **Feed** should be delivered into the ascending buckets, and not into the

boot of the elevator. **Crowned Pulleys** are to be preferred, and the top (driving) pulley may be lagged with a single layer of belting.

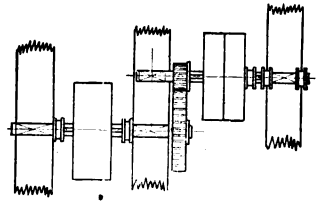


FIG. 7.—Gear drive for elevators.[11]

FEEDING

Feed Control of Ore to Chutes leading to crushing or breaking machinery is regulated by means of gates. Figures 9 and 10 il-

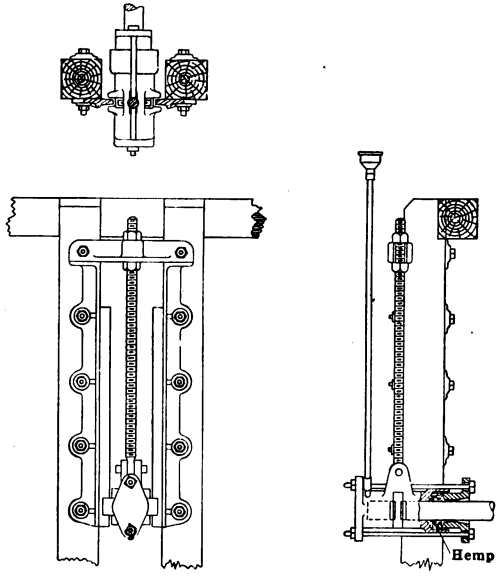
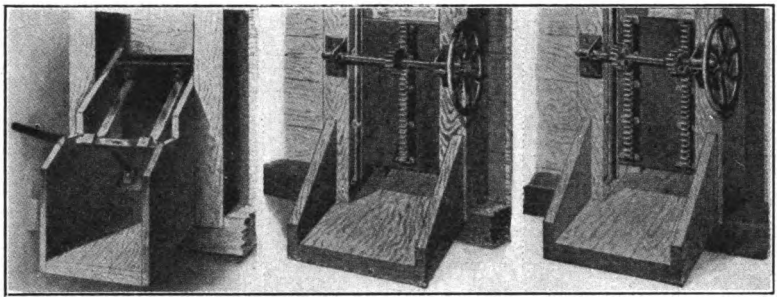


FIG. 8.—Belt-and-bucket elevator take-up boxes.[11]

lustrate the various types in common use. The rack gates are often fitted with counterweights. The sector gate (Fig. 10) is

closed by raising it, by means of a hand lever, through the stream of ore. When full open the back of the gate forms a section of



Lever-arc gate Single-rack gate Double-rack gate

FIG. 9.—Ore-bin gates.[18]

the chute bottom. Leakage of fines is liable to occur with this type because the joints are in the bottom of the chute.

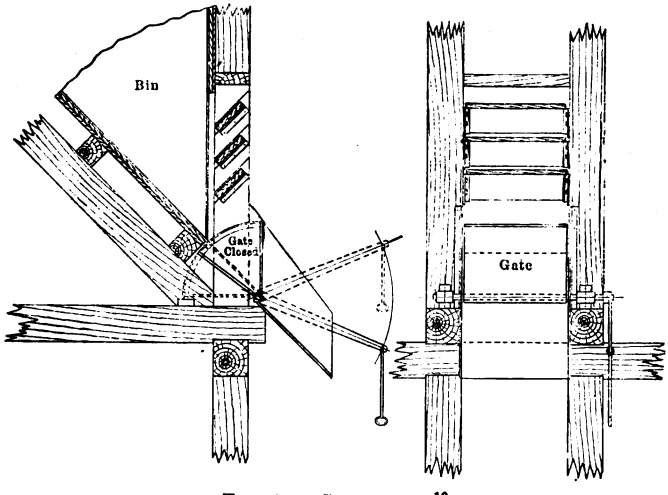


FIG. 10.—Sector gate.[12]

Hand Feeding for Primary Crushers is generally practised in small or medium-sized mills, where it is the usual duty of the

feeder-man to reduce the size of the exceptionally large lumps and also to remove hammer heads, steel, or dynamite which

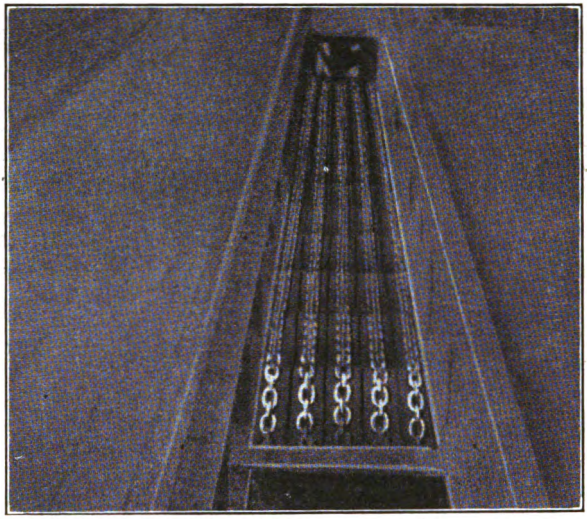


FIG. 11.—Grizzly conveyor.[13]

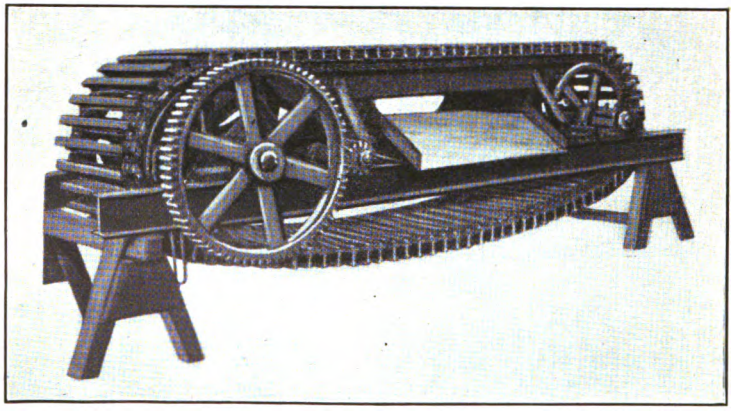


FIG. 12.—Bar-type traveling grizzly.[14]

would cause damage if allowed to remain with the ore. With large mills the problem of primary ore breakage is simplified,

because the capacity of the plant demands the erection of a crusher or crushers of a dimension suitable for the coarsest rock

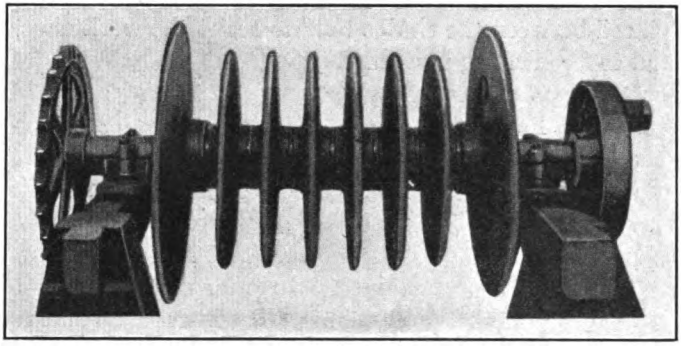


FIG. 13.—Revolving-disc grizzly feeder.[14]

usually received from the mine. Magnets or magnetic pulleys are frequently used to remove steel from crushed ore feed.

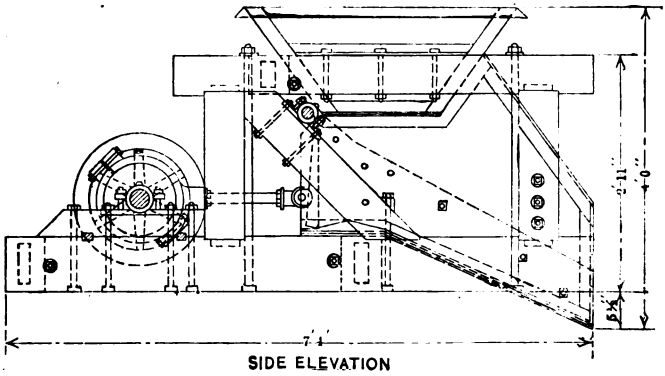


FIG. 14.—Plunger feeder for coarse ore.[15]

The **Chain Grizzly Conveyor** illustrated in Fig. 11 acts as the feeder to a large crusher at the Rowe Mine, Riverton, Minn. The pockets on alternate driving sheaves are arranged so that

adjacent chains travel at different speeds. Masses of agglomerated fine ore are therefore broken up en route to the crusher, the fines falling to the ore bin beneath. Stationary longitudinal bars placed between the chains tend to assist the work of screening, and insure a finer undersize.

The **Bar-Type of Traveling Grizzly** (Fig. 12) is used for feeding crushers. Pieces of ore that become wedged between the bars are automatically released with the oversize, as the bars open up when passing over the sprockets.

The **Revolving-Disc Grizzly Feeder** illustrated in Fig. 13 makes possible the bypassing of fines and the efficient separation of coarse material with a minimum of fall. The feeder illustrated is used in connection with coarse crushers.

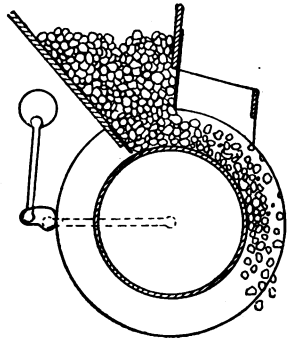


FIG. 15.—Roll feeder.[14]

The **Plunger Feeder for Coarse Ore** (Fig. 14) is suitable for the automatic feeding of mine rock up to 10 in in diameter from chute of storage bin into rock crusher or conveyor. Rate of feed is adjusted by varying the length of the eccentric stroke or by an alteration in the speed.

In **Roll Practice** it is essential that the feed should be evenly distributed over the whole width of the rolls. Short conveyors are sometimes used for this purpose.

Roll Feeders (Fig. 15) are chiefly used for a comparatively fine product. They operate on much the same principle as the Challenge feeder and are fitted with a vertical friction disc. Feeders of this type have also been successfully installed for feeding mine rock to large primary crushers.

Excessive Damage to Belt Conveyors invariably occurs when no feeder is interposed between breaker or bin, and conveyor. The apron type of feeder suitable for this purpose is shown in

Fig. 16. This feeder may also be used to feed crushers from bins containing mine rock.

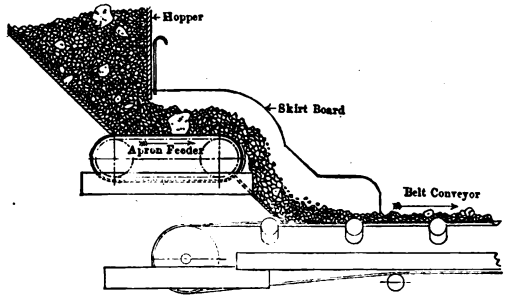


FIG. 16.—Feeder for belt conveyors.[14]

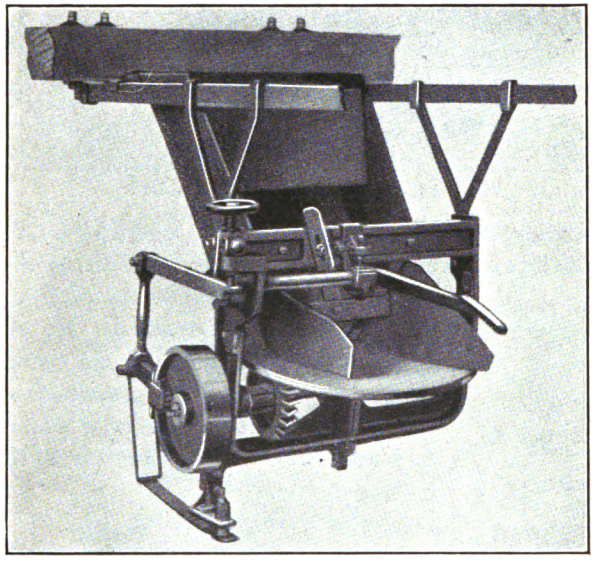


FIG. 17.—Suspended Challenge ore feeder.[11]

Stamp-Mill Feeders of the Challenge Type (Figs. 17, 18, and 19) are largely used, in spite of the inherent defects of design

in the older patterns. Extensive repair and renewal work on feeders of this kind is usually unavoidable.

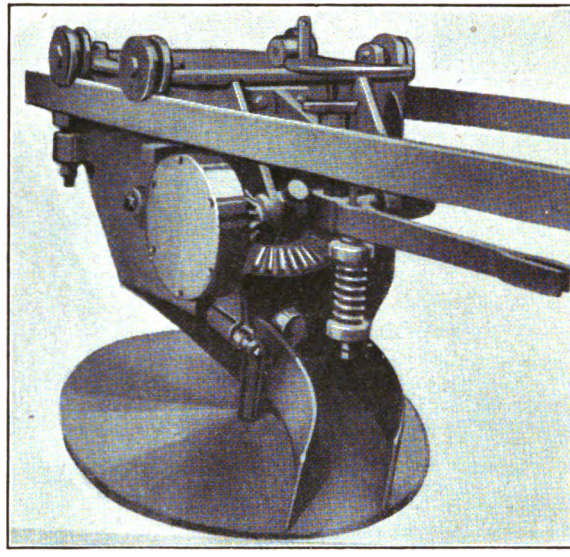


FIG. 18.—Movable Challenge feeder.[16]

The **Monkey-Wrench Feeder** (Fig. 20) operates on the principle of the spanner referred to. The revolving disc and stationary guide, features of the ordinary Challenge type, are retained. No gears are used, and in place of the usual pawl and ratchet a large modified monkey wrench operates on a secondary disc placed below and attached to the main feeder disc.

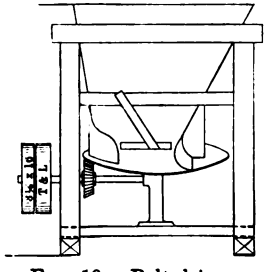


FIG. 19.—Belt-driven Challenge feeder.[17]

The **Plunger Feeder** (Fig. 21) is especially suitable for handling crushed material. It is operated by means of an adjustable eccentric, and can therefore be regulated to suit estimated requirements. This type of feeder is available for any class of crushing or

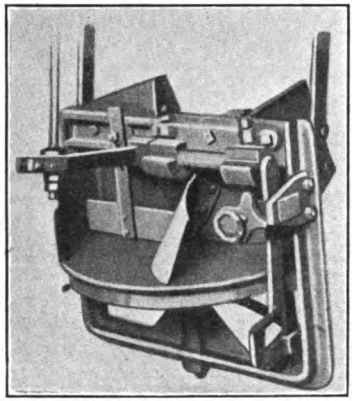


FIG. 20.—Monkey-wrench Challenge feeder.[16]

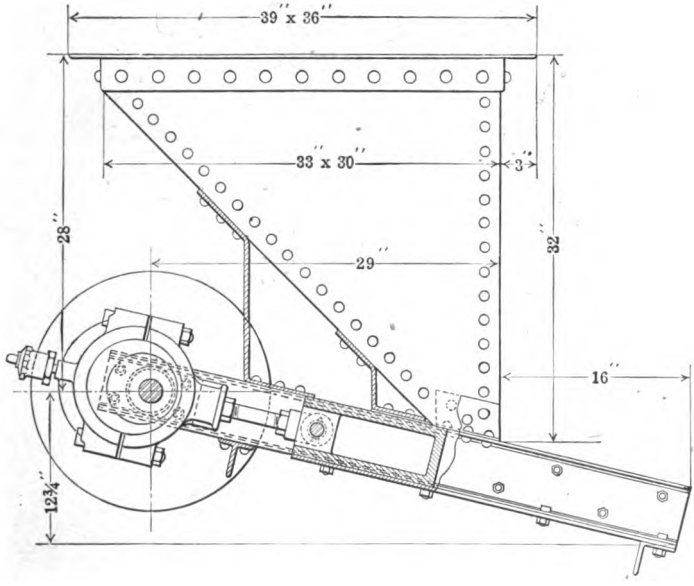


FIG. 21.—Plunger feeder.[18]

milling machine where construction or operation does not permit of automatic adjustment limiting the output from the feeder in accordance with the requirements of the milling machine. A design showing plunger feeder arrangement with direct connection to bin bottom is shown in Fig. 22.

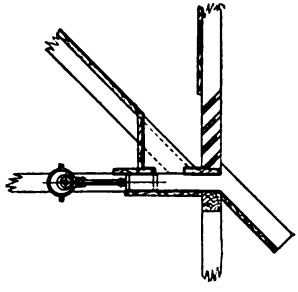


FIG. 22.—Plunger feeder in bin bottom.[19]

An **Automatic Feeder** designed for use with Tremain stamps is shown in Fig. 23. The moving part of the feeder has a surface which sustains but a small part of the weight of ore, the remainder being carried by the hopper. Only a light stroke

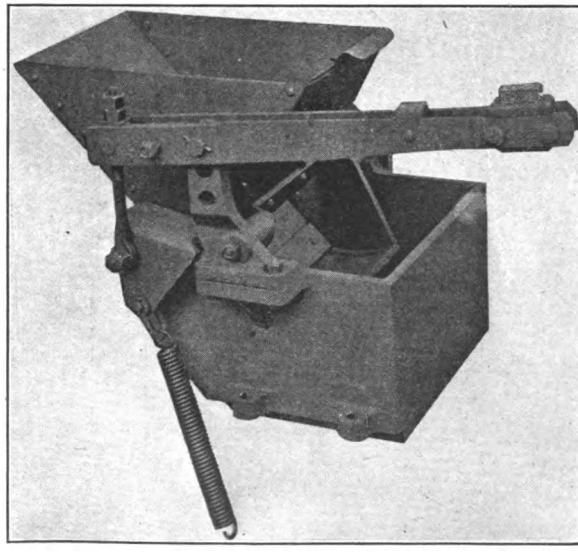


FIG. 23.—Gates automatic feeder.[11]

is necessary, and the feeder is attached directly to the stamp mortar box.

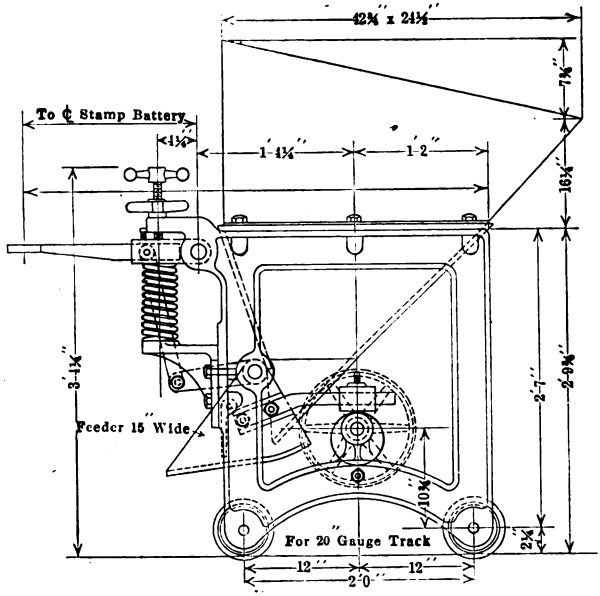


FIG. 24.—Shovel feeder.[19]

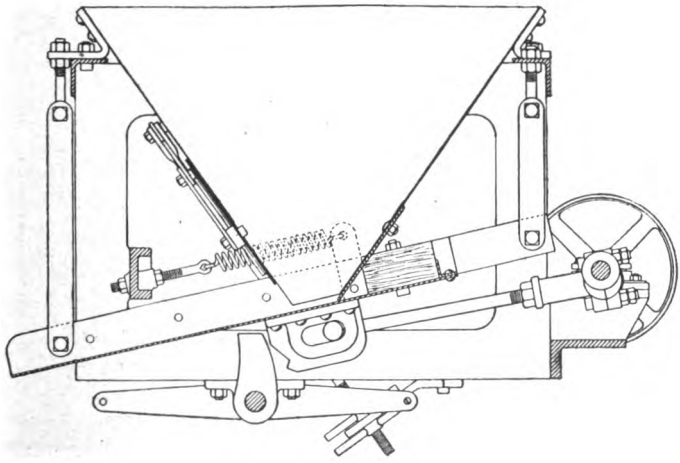


FIG. 25.—Shaking feeder.[12]

In the **Shovel Feeder,** shown in Fig. 24, a projection from the falling stamp throws the boot backward. The counterweight then carries it forward. The combination produces a feeding motion similar to hand shoveling.

The **Shaking Feeder** (Fig. 25) may be adjusted to give an almost continuous flow of crushed ore. This type of feeder is often erected from scrap, and operated by means of a discarded

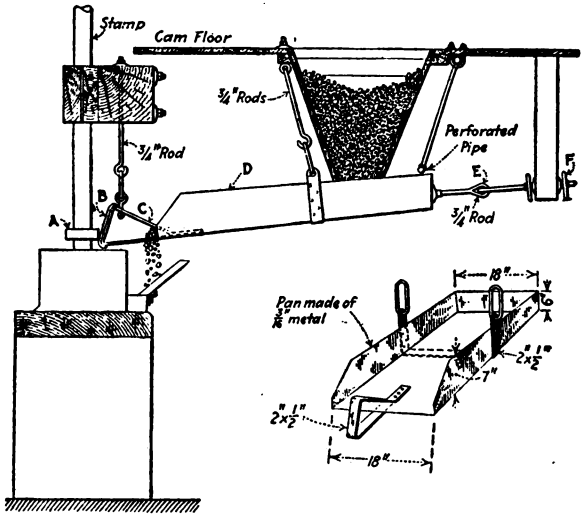


FIG. 26.—Ballow battery feeder.[20]

Wilfley or other concentrator-table head motion. In larger sizes this type has been used as a crusher feeder for coarse ore.

Bumper-Plate Feeders are largely used for the mixed feed delivered to ball mills. West Australian practice gives the following data:

Size of feeder	6 ft. 6 in. long 2 ft. wide at top 1 ft. 3 in. wide at bottom.
Inclination	12 deg. to the horizontal.
Stroke	2 in.
Frequency	41 per minute.

Adjustment of stroke regulates amount of feed.

Bumper-Plate Feeders of the Ballow type (Fig. 26) operate by means of a feeder tappet striking a triangular projection

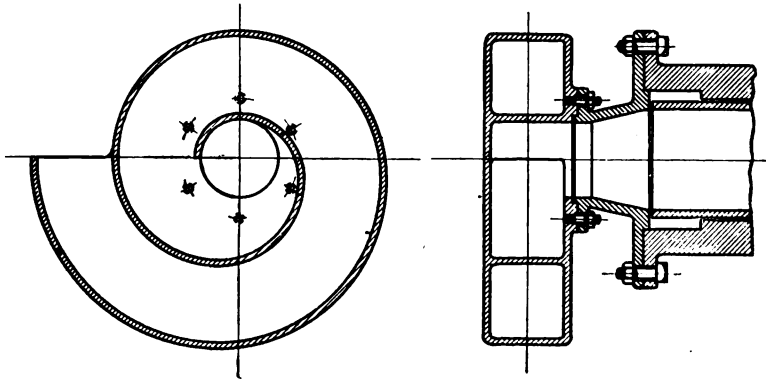


FIG. 27.—Tube-mill feeder.[11]

from a suspended feeder plate. The width of projection engaged by the feeder tappet regulates the jar and, consequently, the feed supply. This type of feeder is especially suitable for

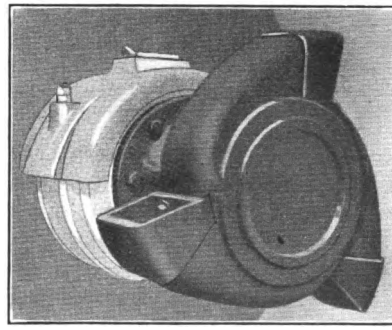


FIG. 28.—Triple spiral feeder.

dry-ore feeding to stamp batteries, is of simple construction, adjustable operation, and requires the minimum of attention and repair.

Tube-Mill Feeders are usually of the spiral type (Fig. 27) and are bolted direct to the trunnion. Ample dimension to allow for the passage of large pebbles should be provided, the cutting edge of the spiral should be faced with a renewable steel plate, and the front of the feeder should have a removable disc permitting examination of interior of the mill as well as of the trunnion liners.

Ball Mills, other than with peripheral discharge, are usually fitted with spiral feeders of the tube-mill pattern, or the type

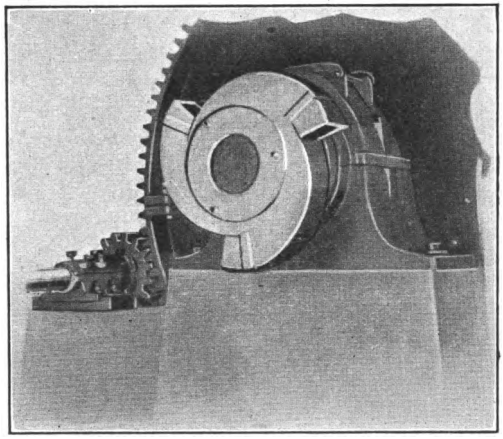


FIG. 29.—Combination triple spiral feeder.

illustrated in Fig. 28. This latter type may also be arranged as a combination feeder in which the pulp may be delivered centrally, and the scoops reserved for feeding the pebbles or balls used as grinding media (Fig. 29). The triple-type spiral feeder is suitable for large-tonnage ball-mill practice.

ORE SAMPLING

No logical attempt can be made to adhere to theoretical limitations in practical ore-sampling work. When ores are being purchased an agreement between buyer and seller in the matter

of sampling is the primary consideration. In all metallurgical plants efficient sampling is an imperatively necessary phase of operations.

All Ore-Sampling Methods must involve bulk reduction by stages. Systematic comminution and thorough mixing are necessary, and both of these steps should preferably be taken before each cutting or splitting operation.

Suitable Weights to be Taken in Sampling Ore are indicated in Table VII. These figures are based on the assumption that the weight taken should be proportional to the square of the diameter of the largest particle.

TABLE VII.—WEIGHTS TO BE TAKEN IN SAMPLING ORES[2]

Weight		Diameter of largest particle					
Grams	Lb.	Very low-grade or very uniform ores, mm.	Low-grade or uniform ores, mm.	Medium ores, mm.		Rich or spotted ores, mm.	Very rich or very spotted ores, mm.
	20,000	207	114	76.2	50.8	31.6	5.4
	10,000	147	80.3	53.9	35.9	22.4	3.8
	5,000	104	56.8	38.1	25.4	15.8	2.7
	2,000	65.6	35.9	24.1	16.1	10.0	1.7
	1,000	46.4	25.4	17.0	11.4	7.1	1.2
	500	32.8	18.0	12.0	8.0	5.0	0.85
	200	20.7	11.4	7.6	5.1	3.2	0.54
	100	14.7	8.0	5.4	3.6	2.2	0.38
	50	10.4	5.7	3.8	2.5	1.6	0.27
	20	6.6	3.6	2.4	1.6	1.0	0.17
	10	4.6	2.5	1.7	1.1	0.71	0.12
	5	3.3	1.8	1.2	0.8	0.5	
	2	2.1	1.1	0.76	0.51	0.32	
	1	1.5	0.8	0.54	0.36	0.22	
	0.5	1.0	0.57	0.38	0.25	0.16	
90	0.2	0.66	0.36	0.24	0.16	0.10	
45	0.1	0.46	0.25	0.17	0.11		
22.5	0.05	0.33	0.18	0.12			
9	0.02	0.21	0.11				
4.5	0.01	0.15					
2.25	0.005	0.10					

A **Sample-Cutting and Reducing Plant** is seldom necessary except where the ore is being purchased on assay content by the milling company, or in the case where subsequent milling operations prohibit the taking of a sample after secondary crushing. In the majority of mills what is generally known as a "battery-feed sample" is taken periodically from the stream of broken ore entering the milling machine. In some cases it has been found possible to hinge a chute, which may be interposed between feeder and mill and across the flow of ore, and so periodically deflect a representative sample for further reduction.

Sampling Practice in Custom Mills varies widely, but it is generally considered necessary to reduce the ore to a uniform size of 2½ in. before any cutting is made. About 20 per cent. is then cut and reduced to 1½ in. A further 20 per cent. is cut from this and reduced to 1 in. An additional 20 per cent. is then cut and reduced to ½ in. The final ½-in. sample is cut according to its bulk, reground to about ⅛ in., again cut, reground to 1/30 in., and finally cut to produce about a 3-lb. sample. The latter is split according to requirements, and further reduced with small equipment in the assay office.

General Mill Sampling is best effected after secondary crushing. The ore is mixed, the flow is more even, and facilities for sampling are generally better. The value of such sampling is, of course, invalid if the ore is being milled in a solvent of one of the metals for the recovery of which the operations are being conducted. Milling gold or silver ores in cyanide solution, and the inevitable admixture, before secondary crushing, of ore, valuable metal in solution, and an active solvent, precludes the possibility of effective sampling after the ore is wetted.

Automatic Ore-Pulp Sampling, as distinct from splitting or dividing, consists of removing a definite proportion of the ore stream either continuously or intermittently, by deflecting the whole stream momentarily, or by passing a cutter across the stream (see Fig. 30).

Automatic Sample Bulk Reduction may be accomplished by means of machines known as "splits" or dividers. These

divide the sample into two or more equal and representative portions, of which one or more may be discarded. The same machines may be used to produce duplicate or triplicate samples. The principle of the Jones sample divider will be understood

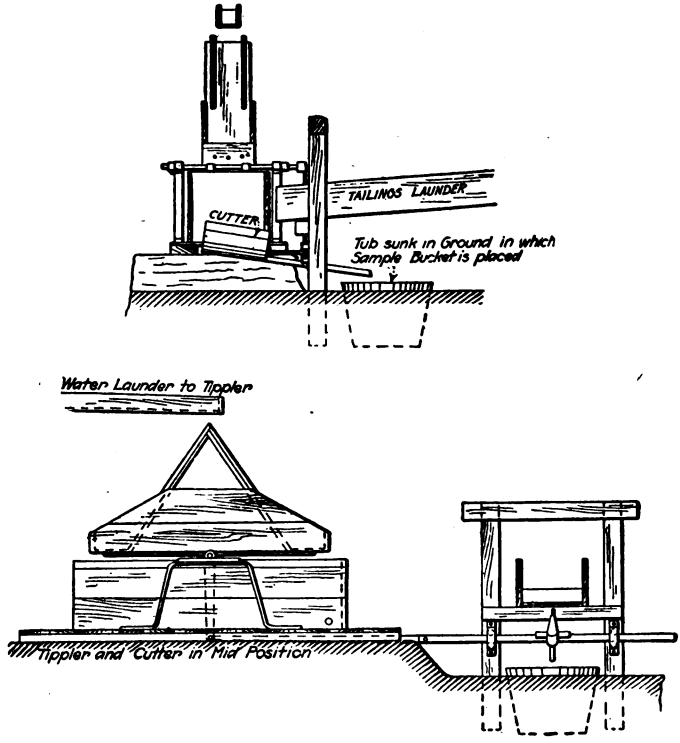

TAILINGS LAUNDER
CUTTER
Tub sunk in Ground in which Sample Bucket is placed
Water Launder to Tippler
Tippler and Cutter in Mid Position

FIG. 30.—Water-actuated pulp sampler.[21]

on reference to Fig. 31. Dimensions are according to size of material and capacity required.

Mass Reduction in Hand Sampling by what is known as **Coning and Quartering** is not essential, neither is it so satisfactory as is generally supposed. The labor involved is considerable, and intelligent operation or skilled supervision is necessary.

Best results are obtained by using a large mounted funnel to form the cone, but even then the apex, of soft material, is liable to collapse at intervals and seldom distributes itself evenly. By shoveling from one spot to another and by rejecting alternate shovelfuls the reduction in bulk may be accomplished satisfactorily with minimum labor and in a much shorter time than by coning and quartering.

Automatic Crushing and Sampling may be carried out in one machine, such as the Clift. In this apparatus

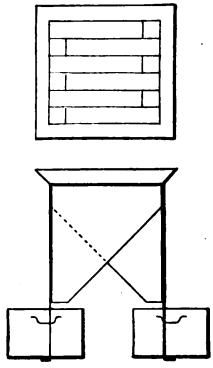


FIG. 31.—Jones split or sample divider.

the stream of ore is divided four times, taking a 20 per cent. cut at each operation, and is crushed three times. The final sample is screened automatically and the oversize is returned for further reduction. A diagram illustrating the operation is shown in Fig. 32. By preceding this apparatus with a large crusher, and by cutting the main stream of ore and sending a proportion to a smaller crusher or rolls, the product from the latter may be sent to the sampling and crushing machine described above.

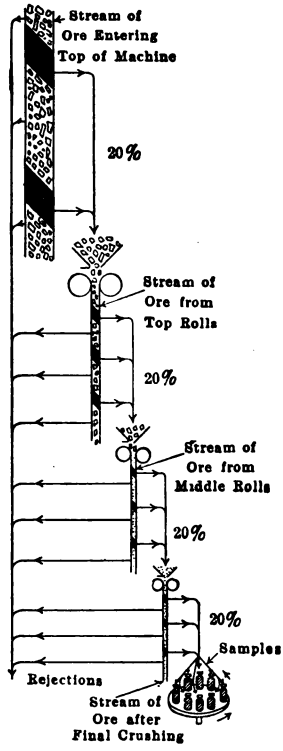
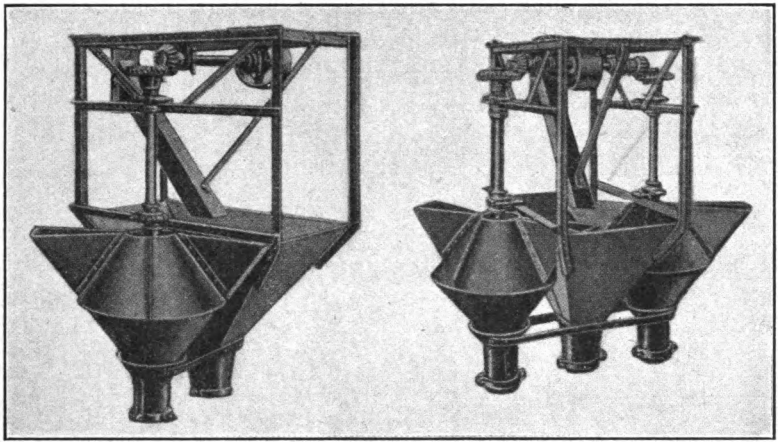


FIG. 32.—Diagram illustrating operation of Clift automatic sampler and crusher.[18]

Single- and Double-Cut Vezin Samplers are illustrated in Fig. 33, from which the principle of the apparatus can be easily understood. Width of cutter opening in relation to maximum-sized particle in ore stream should be ample, and speed should be well below the point tending to hold the ore at the periphery of the cutter by centrifugal force.



Single double-cut Vezin sampler Twin double-cut Vezin sampler

FIG. 33.

Sampling of Sand in situ in leaching or collecting vats is usually done with an auger bit. A clean core may be obtained at selected points provided the sand is sufficiently damp to cohere. Sampling of ore in dumps is done by trenching at intervals across the whole width, and sampling the exposed faces at regular distances apart.

SECTION III

ORE TONNAGE AND MOISTURE ESTIMATION

Weighing of Ore is usually performed in trucks with some type of platform scale. In cases where actual weighing is not practised it is customary to estimate ore weight by calculation of car tally and average. The result is an approximation only. Under- or over-filling of the cars, error in recording numbers, and varia-

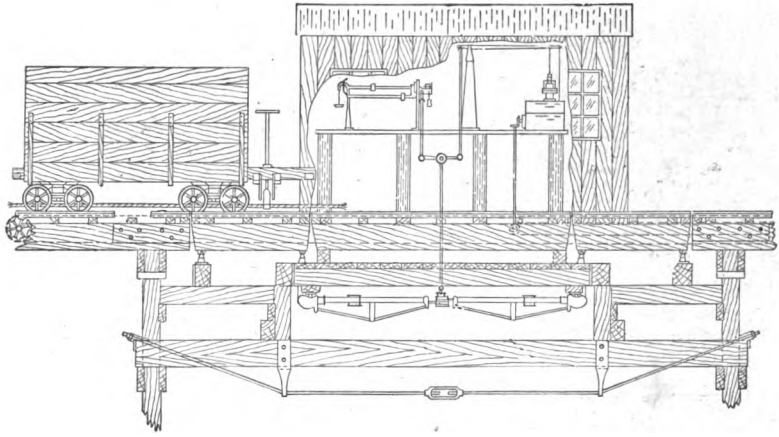


FIG. 34.—Automatic platform weighing scale.[22]

tion in content, resulting in above- or below-normal weights, are among the factors influencing the correctness of result.

The **Automatic Recording of Ore Weight** can be accomplished by means of an apparatus attached to any system of ore conveyance. Figure 34 shows an arrangement automatically to weigh and record weight of ore in cars passing a certain point. Figure 35 gives a sectional view of a machine recording the weight of ore traveling on a belt conveyor. The latter type of apparatus is available for use in connection with bucket, pan, or apron

conveyors, or cable tramways, as well as horizontal or inclined belt conveyors; and an accuracy of 99 per cent. is guaranteed.

The correct **Moisture Percentage** in **Unground Rock** is difficult if not impossible to determine, although commonly attempted. Water occurs in ores as hygroscopic moisture and also as water of crystallization. True ore weight should include the latter but should exclude the former. In estimating moisture it is

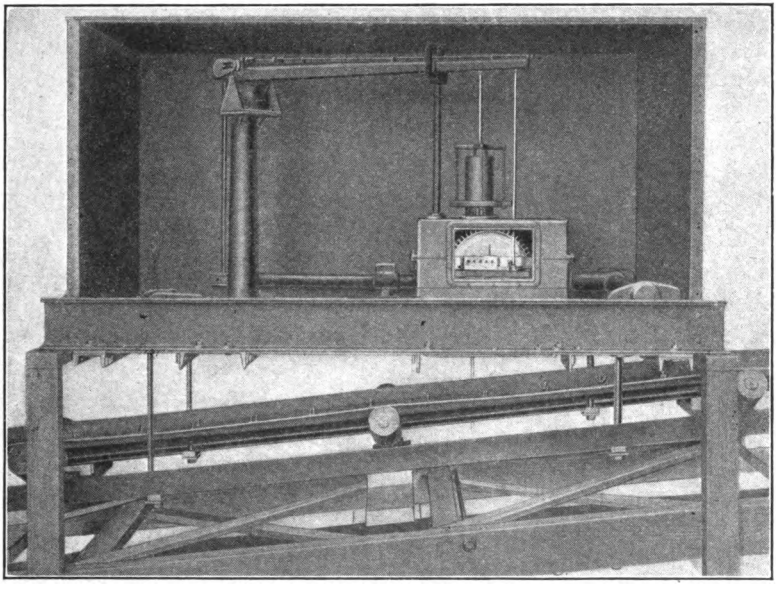


FIG. 35.—Merrick weightometer on belt conveyor. Front sheet of weightometer housing is removed to show interior.[22]

necessary to follow precise laboratory methods. Over-heating may drive off water of crystallization, and under-heating may result in the retention of hygroscopic moisture, and error in statement. Estimation of moisture percentage by weighing before and after heating to an indefinite temperature for an indefinite time results in an approximation only, and should be considered as such.

Estimation of Moisture in Ground Ore is made by selecting a sample without loss of time or undue exposure to the air. An appropriate amount is weighed, and kept at an even temperature of about 110°C. for several hours, and until further heating results in no appreciable reduction in weight. The sample should be cooled in a desiccator before being weighed.

The **Quantity of Material Needed for Moisture Estimation** varies inversely as to the degree of uniformity with which the moisture is distributed. Approximations of moisture content in unground ore are generally made by drying an occasional carload. Moisture content in wet sand or dump tailing is usually calculated from a 100-gm. sample. Pulp samples for tonnage or moisture estimation are generally measured and weighed in a 1000-c.c. flask, and the required figures calculated from tables, without drying the sample.

Clay and Clayey Ores may carry a high percentage of hygroscopic moisture without appearing damp to the sense of touch. A considerable amount of hygroscopic moisture may remain after heating to 100° C., whereas water of crystallization may be expelled and the chemical and physical composition of the mineral may be altered by heating above 110° C.

The **Specific Gravity** of an ore is determined by weighing a portion of the dry, ground material and transferring this to a flask of definite cubic content. The flask is then filled with water to the mark and weighed. Then if:

A = weight of dry ore,
B = weight of flask, ore, and water to mark,
C = weight of flask and water to mark,

then:

specific gravity of the ore = weight of ore ÷ weight of water it displaces $= A/[(C + A) - B]$

The **Estimation** of **Ore-pulp Tonnage** is made, in most instances, by direct weighing and calculation, from a small sample. It is necessary to know the weight of unit volume, or the specific gravity of the pulp, and also the specific gravity of the ore comprising the solid material present. The calculation of dry

tonnage can then be made in several ways by means of the formulæ contained in Table VIII or from the actual equivalents in Table IX.

TABLE VIII.—ORE-PULP TONNAGE FORMULÆ[23]

d = specific gravity of dry solid (ore, sand, or slime).
p = specific gravity of pulp (mixture of water, ore, etc.)
S = per cent. by weight of dry solid in pulp.
= grams in 100 grams, tons in 100 tons, etc.
R = water ratio in pulp.
= tons water per ton dry solid.
= grams water per gram solid.
V = volume per cent. of solid in pulp.
= cubic centimeters in 100 c.c.
= cubic feet in 100 cu. ft. pulp.
F = solid factor.
= grams solid in 100 c.c. pulp.
= tons solid in 100 fluid tons, or 3200 cu. ft. pulp (approx.)
= av. ounces in 0.1 cu. ft. pulp (approx.).
K = constant for any particular solid, used to facilitate calculation and depending on the specific gravity of the dry solid.

VALUES OF K FOR SOLIDS OF KNOWN SPECIFIC GRAVITY

Specific gravity of dry solid d	Values of $K = \frac{100d}{d-1}$	Values of $27K$	Values of $\frac{100}{K} = \frac{d-1}{d}$
2.2	183.3	4950	0.545
2.3	176.9	4780	0.565
2.4	171.4	4630	0.583
2.5	166.7	4500	0.600
2.6	162.5	4390	0.615
2.7	158.8	4290	0.630
2.8	155.5	4200	0.643
2.9	152.6	4120	0.655
3.0	150.0	4050	0.667
3.1	147.6	3985	0.677
3.2	145.5	3930	0.687
3.3	143.5	3875	0.697
3.5	140.0	3780	0.714
4.0	133.3	3600	0.750
5.0	125.0	3375	0.800
6.0	120.0	3240	0.833

TABLE IX.—SLIME DENSITY RELATIONS[24]

Per cent. solids	Ratio of solids to solution	Specific gravity of pulp and volume of one ton in cubic feet, for slimes containing solids of different specific gravities									
		2.50		2.60		2.70		2.80		2.90	
		S. G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.
5	1:19.000	1031	31.03	1032	31.01	1032	30.99	1033	30.97	1034	30.95
6	1:15.667	1037	30.85	1036	30.82	1039	30.79	1040	30.76	1041	30.74
7	1:13.286	1044	30.66	1045	30.62	1046	30.59	1047	30.56	1048	30.53
8	1:11.500	1050	30.46	1052	30.43	1053	30.39	1055	30.36	1055	30.32
9	1:10.111	1057	30.27	1059	30.23	1060	30.19	1061	30.15	1063	30.11
10	1: 9.000	1064	30.08	1065	30.03	1067	29.99	1068	29.95	1070	29.90
11	1: 8.091	1071	29.88	1073	29.83	1074	29.79	1076	29.74	1078	29.69
12	1: 7.333	1078	29.70	1080	29.64	1082	29.59	1083	29.53	1085	29.48
13	1: 6.692	1085	29.50	1087	29.44	1089	29.39	1091	29.33	1093	29.27
14	1: 6.144	1092	29.31	1094	29.24	1097	29.19	1099	29.12	1101	29.06
15	1: 5.667	1099	29.18	1102	29.05	1104	28.99	1107	28.91	1109	28.85
16	1: 5.250	1106	28.93	1109	28.85	1112	28.78	1115	28.71	1117	28.65
17	1: 4.882	1114	28.74	1117	28.65	1119	28.58	1123	28.50	1125	28.44
18	1: 4.556	1121	28.54	1125	28.45	1128	28.38	1131	28.30	1134	28.23
19	1: 4.263	1129	28.35	1133	28.26	1136	28.18	1139	28.09	1142	28.02
20	1: 4.000	1136	28.17	1140	28.06	1144	27.98	1147	27.89	1151	27.81
21	1: 3.762	1144	27.97	1148	27.87	1152	27.77	1156	27.68	1159	27.60
22	1: 3.545	1152	27.78	1157	27.67	1161	27.57	1165	27.47	1168	27.39
23	1: 3.348	1160	27.58	1165	27.47	1169	27.37	1174	27.27	1177	27.18
24	1: 3.167	1168	27.39	1173	27.27	1178	27.17	1182	27.06	1186	26.97
25	1: 3.000	1176	27.21	1182	27.08	1187	26.97	1191	26.85	1195	26.76
26	1: 2.846	1185	27.01	1190	26.88	1195	26.77	1201	26.65	1205	26.55
27	1: 2.704	1193	26.82	1199	26.68	1205	26.56	1210	26.44	1215	26.84
28	1: 2.571	1202	26.62	1209	26.49	1214	26.36	1220	26.24	1224	26.13
29	1: 2.448	1211	26.43	1217	26.29	1223	26.16	1229	26.03	1234	25.92
30	1: 2.333	1220	26.24	1226	26.10	1233	25.95	1239	25.83	1244	25.71
31	1: 2.226	1229	26.05	1236	25.90	1242	25.75	1249	25.63	1255	25.50
32	1: 2.125	1238	25.86	1245	25.70	1252	25.55	1259	25.42	1265	25.29
33	1: 2.030	1247	25.66	1255	25.50	1262	25.35	1269	25.21	1276	25.08
34	1: 1.940	1256	25.47	1264	25.31	1272	25.15	1279	25.01	1287	24.87

35	1: 1.857	1266	25.28	1274	25.12	1283	24.95	1290	24.80	1298	24.66
36	1: 1.778	1276	25.09	1284	24.91	1293	24.75	1301	24.60	1309	24.45
37	1: 1.703	1285	24.90	1295	24.71	1304	24.55	1312	24.39	1320	24.24
38	1: 1.632	1295	24.70	1305	24.52	1314	24.35	1323	24.19	1332	24.03
39	1: 1.564	1305	24.51	1316	24.32	1326	24.14	1335	23.98	1343	23.82
40	1: 1.500	1316	24.32	1326	24.13	1336	23.95	1346	23.77	1355	23.61
41	1: 1.439	1326	24.13	1337	23.93	1348	23.74	1357	23.57	1367	23.40
42	1: 1.381	1337	23.94	1348	23.73	1359	23.55	1370	23.36	1380	23.19
43	1: 1.326	1348	23.74	1359	23.53	1371	23.34	1382	23.16	1392	22.99
44	1: 1.273	1359	23.55	1372	23.33	1383	23.15	1395	22.95	1405	22.78
45	1: 1.222	1370	23.36	1383	23.14	1395	22.94	1407	22.74	1418	22.57
46	1: 1.174	1381	23.17	1395	22.94	1408	22.73	1420	22.54	1432	22.36
47	1: 1.128	1393	22.98	1407	22.75	1420	22.54	1433	22.33	1445	22.15
48	1: 1.083	1404	22.78	1419	22.55	1433	22.33	1446	22.12	1458	21.94
49	1: 1.041	1416	22.59	1431	22.35	1446	22.13	1460	21.92	1473	21.73
50	1: 1.000	1429	22.39	1444	22.15	1460	21.92	1473	21.71	1487	21.52
51	1: 0.961	1441	22.21	1458	21.96	1473	21.72	1488	21.51	1502	21.31
52	1: 0.923	1453	22.02	1471	21.76	1487	21.52	1502	21.30	1517	21.10
53	1: 0.887	1466	21.82	1484	21.56	1501	21.32	1516	21.10	1532	20.89
54	1: 0.852	1479	21.63	1498	21.36	1515	21.12	1532	20.89	1548	20.68
55	1: 0.809	1493	21.44	1512	21.17	1530	20.92	1547	20.69	1564	20.47
56	1: 0.786	1506	21.25	1526	20.97	1545	20.72	1563	20.48	1580	20.26
57	1: 0.754	1520	21.06	1540	20.77	1560	20.51	1579	20.27	1596	20.05
58	1: 0.724	1534	20.86	1555	20.58	1574	20.31	1595	20.07	1613	19.84
59	1: 0.695	1548	20.67	1572	20.38	1591	20.11	1611	19.86	1629	19.63
60	1: 0.667	1563	20.48	1585	20.18	1607	19.91	1628	19.66	1645	19.42
61	1: 0.639	1577	20.29	1601	19.98	1623	19.71	1645	19.45	1664	19.21
62	1: 0.613	1592	20.10	1617	19.79	1641	19.51	1662	19.25	1683	19.00
63	1: 0.587	1608	19.90	1633	19.59	1657	19.30	1681	19.04	1703	18.79
64	1: 0.563	1623	19.71	1650	19.40	1675	19.10	1698	18.94	1723	18.58
65	1: 0.538	1639	19.52	1667	19.20	1692	18.90	1718	18.73	1742	18.37
66	1: 0.515	1656	19.32	1684	19.00	1711	18.70	1738	18.53	1762	18.16
67	1: 0.493	1672	19.14	1701	18.80	1730	18.50	1757	18.32	1783	17.95
68	1: 0.471	1689	18.94	1719	18.61	1749	18.30	1776	18.11	1803	17.74
69	1: 0.449	1706	18.75	1738	18.41	1768	18.10	1797	17.81	1825	17.53
70	1: 0.429	1724	18.56	1757	18.21	1786	17.90	1818	17.60	1847	17.32

TABLE IX.—SLIME DENSITY RELATIONS (*Continued*)

Per cent. solids	Ratio of solids to solution	Specific gravity of pulp and volume of one ton in cubic feet, for slimes containing solids of different specific gravities									
		3.00		3.10		3.20		3.30		*4.50	
		S. G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.
5	1:19.000	1035	30.93	1035	30.92	1036	30.90	1036	30.89	1040	30.76
6	1:15.667	1042	30.72	1042	30.70	1043	30.68	1043	30.66	1049	30.51
7	1:13.286	1049	30.51	1049	30.48	1050	30.46	1051	30.43	1058	30.26
8	1:11.500	1056	30.30	1057	30.27	1058	30.24	1059	30.21	1067	30.01
9	1:10.111	1064	30.09	1065	30.05	1066	30.02	1067	29.99	1075	29.76
10	1: 9.000	1071	29.87	1072	29.83	1074	29.80	1075	29.77	1084	29.51
11	1: 8.091	1078	29.65	1080	29.61	1082	29.58	1083	29.54	1093	29.26
12	1: 7.333	1087	29.44	1088	29.40	1090	29.36	1091	29.32	1102	29.01
13	1: 6.692	1095	29.23	1096	29.18	1098	29.14	1099	29.10	1112	28.76
14	1: 6.144	1103	29.01	1105	28.96	1106	28.92	1108	28.88	1122	28.52
15	1: 5.667	1111	28.80	1113	28.74	1115	28.70	1117	28.66	1132	28.27
16	1: 5.250	1119	28.59	1122	28.53	1124	28.48	1125	28.43	1142	28.02
17	1: 4.882	1128	28.37	1130	28.31	1132	28.26	1134	28.21	1152	27.77
18	1: 4.556	1136	28.16	1139	28.10	1141	28.04	1143	27.99	1163	27.52
19	1: 4.263	1145	27.95	1148	27.88	1150	27.82	1153	27.76	1173	27.27
20	1: 4.000	1154	27.73	1157	27.66	1159	27.60	1162	27.54	1184	27.02
21	1: 3.762	1163	27.52	1166	27.44	1169	27.38	1171	27.32	1194	26.77
22	1: 3.545	1172	27.31	1175	27.23	1178	27.16	1181	27.09	1206	26.52
23	1: 3.348	1181	27.09	1184	27.01	1188	26.94	1191	26.87	1218	26.28
24	1: 3.167	1190	26.88	1194	26.79	1198	26.72	1201	26.65	1230	26.03
25	1: 3.000	1200	26.67	1204	26.58	1208	26.50	1211	26.42	1241	25.78
26	1: 2.846	1210	26.45	1214	26.37	1218	26.28	1222	26.20	1253	25.53
27	1: 2.704	1220	26.24	1224	26.15	1228	26.06	1232	25.98	1266	25.28
28	1: 2.571	1230	26.03	1234	25.93	1239	25.84	1242	25.75	1278	25.03
29	1: 2.448	1240	25.81	1244	25.71	1249	25.62	1253	25.53	1291	24.78
30	1: 2.333	1250	25.60	1255	25.50	1260	25.40	1264	25.31	1304	24.53
31	1: 2.226	1261	25.39	1266	25.28	1271	25.18	1275	25.08	1317	24.28
32	1: 2.125	1271	25.17	1277	25.06	1282	24.96	1287	24.86	1331	24.04
33	1: 2.030	1282	24.96	1288	24.85	1293	24.74	1299	24.64	1345	23.79
34	1: 1.940	1293	24.75	1299	24.63	1305	24.52	1311	24.41	1359	23.54

* 80 per cent. pyrite and 20 per cent. quartz.

35	1: 1.857	1304	24.53	1310	24.41	1317	24.30	1323	24.19	1374	23.29
36	1: 1.778	1316	24.32	1322	24.19	1329	24.08	1335	23.97	1389	23.04
37	1: 1.703	1328	24.11	1334	23.98	1341	23.86	1347	23.75	1404	22.79
38	1: 1.632	1340	23.89	1346	23.76	1353	23.64	1360	23.52	1420	22.54
39	1: 1.564	1351	23.68	1358	23.55	1366	23.42	1373	23.30	1435	22.29
40	1: 1.500	1363	23.47	1371	23.33	1379	23.20	1387	23.08	1451	22.04
41	1: 1.439	1376	23.26	1384	23.11	1393	22.98	1400	22.85	1468	21.79
42	1: 1.381	1389	23.04	1397	22.89	1406	22.76	1414	22.63	1485	21.55
43	1: 1.326	1402	22.83	1411	22.68	1419	22.54	1428	22.41	1502	21.30
44	1: 1.273	1415	22.61	1425	22.46	1433	22.32	1442	22.18	1519	21.05
45	1: 1.222	1429	22.40	1438	22.24	1447	22.10	1456	21.96	1538	20.80
46	1: 1.174	1443	22.19	1452	22.02	1462	21.88	1471	21.74	1557	20.55
47	1: 1.128	1457	21.97	1467	21.81	1477	21.66	1487	21.51	1576	20.30
48	1: 1.083	1471	21.76	1483	21.60	1493	21.44	1503	21.29	1495	20.05
49	1: 1.041	1485	21.55	1497	21.38	1508	21.22	1519	21.07	1615	19.81
50	1: 1.000	1500	21.33	1512	21.16	1524	21.00	1535	20.85	1637	19.56
51	1: 0.961	1515	21.12	1528	20.94	1540	20.78	1551	20.62	1658	19.31
52	1: 0.923	1531	20.91	1544	20.73	1556	20.56	1568	20.40	1679	19.06
53	1: 0.887	1547	20.69	1560	20.51	1573	20.34	1585	20.18	1700	18.81
54	1: 0.852	1563	20.48	1577	20.29	1590	20.12	1603	19.96	1724	17.56
55	1: 0.809	1579	20.27	1594	20.08	1608	19.90	1621	19.73	1748	18.31
56	1: 0.786	1596	20.05	1611	19.87	1626	19.68	1640	19.51	1772	18.06
57	1: 0.754	1613	19.84	1628	19.65	1645	19.46	1659	19.29	1796	17.81
58	1: 0.724	1631	19.63	1646	19.43	1663	19.24	1678	19.06	1822	17.56
59	1: 0.695	1649	19.41	1665	19.21	1682	19.02	1697	18.84	1848	17.32
60	1: 0.667	1667	19.20	1684	19.00	1702	18.80	1718	18.62	1875	17.07
61	1: 0.639	1686	18.99	1704	18.78	1722	18.58	1739	18.39	1903	16.82
62	1: 0.613	1705	18.77	1724	18.56	1742	18.36	1761	18.17	1932	16.57
63	1: 0.587	1724	18.56	1745	18.34	1764	18.14	1783	17.95	1961	16.32
64	1: 0.563	1745	18.35	1765	18.12	1786	17.92	1805	17.72	1992	16.07
65	1: 0.538	1765	18.13	1786	17.91	1808	17.70	1828	17.50	2023	15.82
66	1: 0.515	1786	17.92	1808	17.69	1830	17.48	1852	17.28	2054	15.57
67	1: 0.493	1808	17.71	1831	17.47	1853	17.26	1876	17.06	2088	15.32
68	1: 0.471	1830	17.49	1854	17.26	1877	17.04	1901	16.83	2123	15.08
69	1: 0.449	1852	17.28	1878	17.04	1902	16.82	1927	16.61	2159	14.83
70	1: 0.429	1875	17.07	1902	16.83	1926	16.60	1953	16.39	2195	14.58

$$K = \frac{100d}{d-1}.$$

$$d = \frac{p}{1 - R(p-1)} = \frac{Sp}{Sp - 100(p-1)}.$$

$$p = \frac{R+1}{R+\frac{1}{d}} = \frac{100}{100 - \frac{S(d-1)}{d}} = \frac{K}{K-S}.$$

$$S = \frac{100}{R+1} = \frac{100d(p-1)}{p(d-1)} = \frac{K(p-1)}{p}$$

$$R = \frac{d-p}{d(p-1)} = \frac{1-\frac{d}{p}}{p-1} = \frac{100-S}{S} = \frac{100}{S} - 1 = \frac{100p}{K(p-1)} - 1.$$

$$F = Sp = \frac{100p}{R+1} = \frac{100d(p-1)}{d-1} = K(p-1).$$

$$V = \frac{F}{d} = \frac{Sp}{d} = \frac{100(p-1)}{d-1} = (K-100)(p-1).$$

Volume percentage of water in pulp $= 100 - V = 100 - \frac{F}{d} = p(100-S) = \frac{100(d-p)}{d-1}$.

Tons dry solid per 100 tons water $= \frac{100}{R} = \frac{100S}{100-S} = \frac{100d(p-1)}{d-p}$.

Fluid tons to yield 1 ton solid $= \frac{100}{F} = \frac{100}{K(p-1)} = \frac{R+1}{p}$.

Sand Tonnage in situ in a leaching or collecting vat is calculated by allowing a box of 1-cu. ft. capacity to fill with the vat. After drainage of excess moisture, the box and contents are weighed, preferably in the vat by means of a steelyard. A sample of the sand is reserved for immediate moisture determination, a deduction is also made for the weight of the box, and the vat contents are finally calculated from the net weight of the cubic foot of sand and the volume of the vat as filled.

SECTION IV

THEORY OF CRUSHING

The **Statement of Crushing Result** in terms of mechanical values must be based on some law defining the ratio between work required or done and reduction in dimension resulting. Much controversy on this point has occurred, and two laws, those of Kick and Rittinger, have been the subject of criticism or support.

Rittinger's Law states that the work of crushing is proportional to the reduction in diameter resulting. Richards' interpretation is identical, but may be stated differently, viz.: that the work done is proportional to the surface exposed. By referring to Fig. 36, representing a 1-in. cube, it will be seen that the surface area is $1[6(1 \times 1)] = 6$ sq. in. If this cube is divided into ½-in. cubes by three planes the surface area will be increased to $8[6(\frac{1}{2} \times \frac{1}{2})] = 12$ sq. in. If reduced to ¼-in. cubes the surface area will be increased to $64[6(\frac{1}{4} \times \frac{1}{4})] = 24$ sq. in. The surface areas of the cubes therefore increase in the proportions 6 : 12 : 24 or 1 : 2 : 4, which corresponds to the reciprocals of the lengths of cube sides, or diameters.

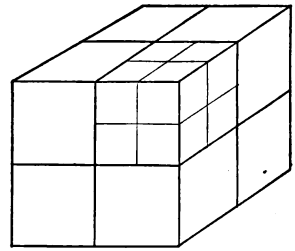


FIG. 36.—Subdivision of cube.

The Relative Surface Exposed in Each Grade May Be Determined by dividing the weight of material found in that grade by the average diameter of the particles. This is based on the fact that the total surface area for any definite weight of ore varies inversely as the diameter of the particles composing that weight.

Average Size of Material in each grade is determined by finding the arithmetical mean width of the apertures in the screen on which the particles remain and the screen of next larger aperture width. This gives a figure which is as approximately correct as possible where screens with close-order difference in aperture width are used.

The Reciprocals of the Mean Diameter of screen aperture width are used by Del Mar in the computation of crushing efficiency comparisons. These reciprocals are multiplied by percentage of material found in the various grades, and the product gives relative surface exposed. By assuming relative surface units and work units as of interchangeable value the comparison may be made in the manner shown in Table X. **Relative Mechanical Efficiencies** may be obtained by multiplying units of work done in each case by duty or output and dividing the result by power consumption.

TABLE X.—COMPARISON OF CRUSHING PERFORMANCE BASED ON RITTINGER'S LAW[25]

	Reciprocals of Sizes of Screen Apertures						
Mesh	Aperture, in.	Reciprocals	Mesh	Mean aperture of grade, in.	Reciprocals	Mean aperture of grade, in.	Reciprocals
20	0.0335	29.8	+ 20	0.0376	26.5	0.0376	26.5
30	0.0195	51.0	+ 30	0.0265	37.7		
40	0.0147	68.0	+ 40	0.0171	58.4	0.0241	41.4
50	0.011	91.0	+ 50	0.01285	77.7		
60	0.0091	110.0	+ 60	0.01005	99.5	0.0119	83.6
80	0.00675	148.0	+ 80	0.00792	138.0	0.00792	138.0
100	0.0055	182.0	+100	0.00612	163.0	0.00612	163.0
			−100	0.0049	204.0	0.0049	204.0
120	0.0043	232.5	+120	0.0049	204.0	0.0049	204.0
			−120	0.00395	253.0	0.00395	253.0
150	0.0036	277.7	+150	0.00395	253.0	0.00395	253.0
			−150	0.0033	303.0	0.0033	303.0
200	0.003	333.3	+200	0.0033	303.0	0.0033	303.0
			−200	0.0025	400.0	0.0025	400.0

TABLE X.—(*Continued*)

Screen Analyses of Mill Material

Mesh	Mill No. 1		Mill No. 2	
	Mill feed, per cent.	Discharge, per cent.	Mill feed, per cent.	Discharge, per cent.
On 20	3.1	0.0	15.1	3.5
On 40	15.5	3.2	35.6	21.2
On 60	15.6	4.5	10.4	14.1
On 80	14.6	10.5	11.1	17.9
On 100	9.6	15.8	6.4	11.2
On 120	2.4	2.4	0.7	1.6
Through 120	39.2	63.6	20.7	30.5

Relative Work Done in Mill No. 1

Mesh	Reciprocal of average size	Feed, per cent.	Relative surface in feed	Discharge, per cent.	Relative surface in discharge
+ 20	26.5	3.1	82	0.0	
+ 40	41.4	15.5	641	3.2	132
+ 60	83.6	15.6	1,304	4.5	376
+ 80	138.0	14.6	2,015	10.5	1,449
+100	163.0	9.6	1,564	15.8	2,575
+120	204.0	2.4	489	2.4	489
−120	253.0	39.2	9,917	63.6	16,091
Units of work in feed			16,012	Units of work in discharge	21,112
Units of work in feed					16,012
Difference, units of work done per ton					5,100

TABLE X.—(*Continued*)

Relative Work Done in Mill No. 2

Mesh	Reciprocal of average size	Feed, per cent.	Relative surface in feed	Discharge, per cent.	Relative surface in discharge
+ 20	26.5	15.1	400	3.5	92
+ 40	41.4	35.6	1,473	21.2	877
+ 60	83.6	10.4	919	14.1	1,178
+ 80	138.0	11.1	1,532	17.9	2,470
+100	163.0	6.4	1,043	11.2	1,825
+120	204.0	0.7	143	1.6	326
−120	253.0	20.7	5,237	30.5	7,716
Units of work in feed			10,747	Units of work in discharge	14,484
Units of work in feed					10,747
Difference, units of work done per ton					3,737

The **Crushing Efficiencies of Various Milling Machines,** calculated on the basis of increased surface exposed during crushing, is given in Table XI.

TABLE XI.—CRUSHING EFFICIENCIES OF VARIOUS MACHINES[26]

Plant	Machine	Relative surface in feed	Relative surface in product	Index of work done by machine
Goldfield Con	Stamps	100	14,007	13,907
Dome	Stamps	100	18,036	17,936
Goldfield Con	High-speed Chilean	720	27,239	26,519
Mogul	High-speed Chilean	100	22,835	22,735
Gold Belt	Slow-speed Chilean	100	32,913	32,813
Sta. Elena	Slow-speed Chilean	75	35,640	35,565
La Union	Slow-speed Chilean	100	37,140	37,040
Llano	Slow-speed Chilean	100	33,108	33,008
Bolivia	Wheeler pan	18,363	36,678	18,315

Rittinger's Law was disproved by Van Reytt as a result of actual demonstration. The tests made showed that the ratio of work done to increased surface is fairly constant with coarse sizes, but with fine sizes the increase of surface is much more rapid than the work required to produce it.

Kick's Law states that the work of crushing is proportional to the reduction in volume resulting. An elaboration of this definition is as follows: The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies. Stadler's suggestions for the adaptation of this law to comparative efficiency estimations are as follows:

(*a*) Classification into grades on the basis of volume or weight.

(*b*) By reducing the cube of the unit successively to one-half its volume, and assuming these fractures to be again of cubical shape, each size of this series of theoretical cubes obtained represents a grade of a reduction scale of the ratio $\sqrt[3]{2}$.

(*c*) If to each of these grades is given an ordinal number, beginning with 0 as representing the unit of 1-in. cube, then Table XII gives figures to determine the values of the data for the series of the theoretical cubes.

(*d*) By taking the sides of this series of cubes as clear-mesh apertures of a set of screens it is admissible to assume that the functions of the irregularly shaped average particles, determined by two successive screens, vary in the same ratio as the grades of the theoretical cubes.

To Obtain the Mechanical Value of a Mixed Product, for comparative purposes, it is necessary to multiply the screen percentages by the number of the energy units of the respective grades and add the products.

The **Relative Mechanical Efficiency** of two machines or of the same machine operating under different conditions may be obtained by dividing the total work done in each instance, as estimated by the energy-unit system, by any convenient unit of energy such as horsepower consumed. The useful work done by the machine may be estimated from the difference between

TABLE XII.—STADLER'S STANDARD GRADES[86]

Ordinal number or mechanical value Energy Units	Mesh aperture (= length of side of theoretical cubes)	
	In.	Mm.
30	0.00098	0.02480
29	0.00123	0.03125
28	0.00155	0.03937
27	0.00195	0.04961
26	0.00246	0.06250
25	0.00310	0.07875
24	0.00391	0.09922
23	0.00492	0.1250
22	0.00620	0.15750
21	0.00781	0.19844
20	0.00984	0.250
19	0.01240	0.3150
18	0.01562	0.39688
17	0.01969	0.50
16	0.0248	0.630
15	0.03125	0.7938
14	0.03937	1.0
13	0.04961	1.260
12	0.06250	1.5875
11	0.07875	2.0
10	0.09922	2.520
9	0.1250	3.1750
8	0.15750	4.0003
7	0.19850	5.040
6	0.250	6.350
5	0.3150	8.0005
4	0.39690	10.080
3	0.50	12.70
2	0.630	16.0
1	0.79370	20.160
0	1.0	25.4

the mechanical values of samples from feed and discharge. An example of this method is given in Table XIII. In this case a comparison was being made between the operation of four Nissen stamps and two five-stamp mills. The drop in each case, both with regard to height and frequency, was identical, and the unit of energy was therefore based on effective weight of stamp. The feed in each case was also identical and was assumed to have a relative mechanical value of zero in comparison with the crushed product.

TABLE XIII.—EXAMPLE OF METHOD OF COMPUTING RELATIVE MECHANICAL EFFICIENCY OF CRUSHING MACHINES[27]

(Based on Kick's Law)

Screening			Four Nissen stamps			Two 5-stamp mills		
Aperture width, in.	Mesh	Approx E. U.	Grading, per cent.		Mech. value, E. U.	Grading, per cent.		Mech. value, E. U.
0.159	+ 5	9	19.80		1.782	14.8		1.332
0.097	+ 8	11	10.75		1.183	11.28		1.241
0.060	+ 12	13	9.30	66.54	1.209	10.59	64.60	1.377
0.034	+ 20	15	10.04		1.506	10.57		1.586
0.0198	+ 30	17	7.92		1.346	8.50		1.445
0.0110	+ 50	19	8.73		1.659	8.86		1.683
0.0068	+ 80	21	10.17	10.17	2.136	11.48	11.48	2.411
0.00433	+120	24	4.71	23.29	1.130	5.38	23.74	1.291
0.0030	+200							
	−200	28	18.58		5.202	18.36		5.141
			100.00	100.00	17.153	99.82	99.82	17.507
Duty per stamp per 24 hours....			27.72 tons			19.9 tons		
Weight of stamp.....			1927 lb.			1860 lb.		
Relative mechanical efficiency....			$\frac{17.153 \text{ E. U.} \times 27.72}{1927} = 0.25$			$\frac{17.507 \text{ E. U.} \times 19.9}{1860} = 0.19$		

Approximate Mechanical Values corresponding to any size of screen aperture may be found from Fig. 37. The top figures (12–26) represent the energy units. Screen aperture widths are given in the left-hand column. The junction of the horizontal

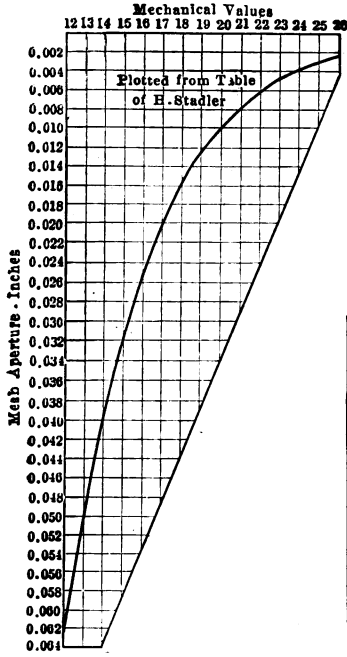
TABLE XIV.—CONSTANTS FOR TYLER SCREEN-SCALE SIEVES[29]

(Graded by Aperture Widths)

Apertures		Average diameter of product		Volumes		Area of fracture		Number of pieces from unit	Energy number, Stadler
Inches	Millimeters	Inches	Millimeters	Cubic inches	Cubic millimeters	Square inches	Square millimeters		
1.050	26.67								
0.742	18.85	0.896	22.76	0.71932	11,790.0	0.0	0.0	1.0	0
0.525	13.33	0.634	16.09	0.25484	4,165.5	0.99529	642.1	2.8826	1
0.371	9.423	0.448	11.377	0.089915	1,472.6	2.4085	1,553.8	8.0000	2
0.263	6.680	0.317	8.052	0.031855	522.05	4.399	2,838.0	22.581	3
0.185	4.699	0.224	5.690	0.011239	184.22	7.2253	4,661.4	64.00	4
0.131	3.327	0.158	4.013	0.0039443	64.626	11.280	7,277.2	182.37	5
0.093	2.362	0.112	2.845	0.0014049	23.028	16.859	10,877.0	512.0	6
0.065	1.666	0.079	2.014	0.00049304	8.1692	24.908	16,069.0	1,459.9	7
0.046	1.178	0.056	1.422	0.00017562	2.8754	36.126	23,307.0	4,096.0	8
0.0328	0.833	0.0394	1.000	0.000061163	1.0000	52.362	33,781.0	11,761.0	9
0.0232	0.589	0.0280	0.711	0.000021952	0.35943	74.662	48,167.0	32,768.0	10
0.0164	0.417	0.0198	0.503	0.0000077624	0.12726	106.58	68,759.0	92,668.0	11
0.0116	0.295	0.0140	0.356	0.000002744	0.045118	151.73	97,889.0	262,140.0	12
0.0082	0.208	0.0099	0.252	0.00000097030	0.016003	215.57	139,070.0	741,340.0	13
0.0058	0.147	0.0070	0.178	0.000000343	0.0056398	305.87	197,330.0	2,097,200.0	14
0.0041	0.104	0.0050	0.126	0.000000125	0.0020004	429.19	276,880.0	5,754,600.0	15
0.0029	0.074	0.0035	0.089	0.000000042875	0.00070497	614.15	396,220.0	16,777,000.0	16
0.0014*	0.037	0.0022	0.056	0.000000010648	0.00017562	978.49	631,260.0	67,555,000.0	17.34

* Assumed.

line from the given screen aperture with the curve shown will give a point vertically below the required mechanical value.



Mechanical Values E.U.	Aperture	
	Width: Inch	Mesh:
1	0.7937	$\frac{3}{4}''$
2	0.63	$\frac{5}{8}''$
3	0.50	$\frac{1}{2}''$
4	0.3969	$\frac{3}{8}''$
5	0.315	$\frac{5}{16}''$
6	0.250	$\frac{1}{4}''$
7	0.1985	3.5
8	0.1575	4
9	0.125	6
10	0.09922	8
11	0.07875	10

FIG. 37.—Stadler energy-unit curve.[28]

A Table of Constants for **Tyler Standard Sieves** calculated on the basis of average sizes of successive screened products rather than aperture sizes, is given in Table XIV. The figures are, therefore, based on average size of product through the coarser sieve on the next finer.

SECTION V

ROASTING

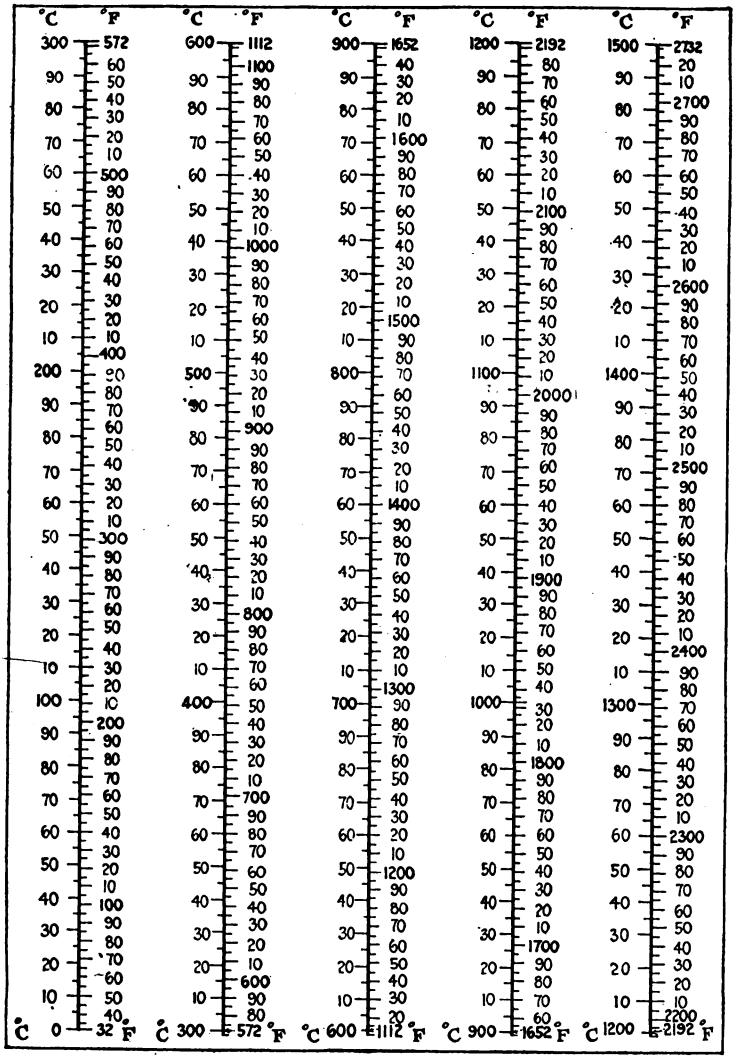
Roasting of Metalliferous Ores is carried out for the purpose of removing certain elements which would interfere with subsequent treatment. Copper, lead, zinc, and gold ores and concentrates are roasted to remove sulphur and in order to make the material amenable to smelting operations, and for the purpose of removing certain volatile refractory metals, such as arsenic and bismuth. Gold ores are roasted as a preliminary to cyanide treatment in order to eliminate the possible action of interfering elements, to reduce tonnage, and to facilitate subsequent grinding. The freeing of gold from encasing pyrite, by roasting, is also an important factor in the amalgamation treatment of a complex ore.

Heap Roasting is a simple process in which a large cone of coarse ore is piled over a stack of wood fuel. The latter when ignited supplies the necessary heat and draught, and the roasting proceeds without additional fuel provided the ore carries sufficient sulphur to maintain combustion.

A chart for the rapid **Conversion of Fahrenheit into Centigrade Temperatures,** or vice versa, is given in Table XV.

The **Edwards Furnace** (Fig. 38), designed in Australia, and used with success in the roasting of the sulpho-telluride ores at Kalgoorlie, is a reverberatory furnace of the straight-line, single-hearth type. There are two fireplaces. The rabbles and rabble spindles are water cooled at the front end of the furnace. These are of the usual revolving type and driven by suitable gearing from the top of the furnace. Inclination of the furnace from the horizontal will depend on the character of the ore and the time required for roasting. Average practice would indicate a fall of about 8 in. from feed to discharge, a distance of 70 ft.

TABLE XV.—CONVERSION SCALE FOR CENTIGRADE AND FAHRENHEIT TEMPERATURE [30]



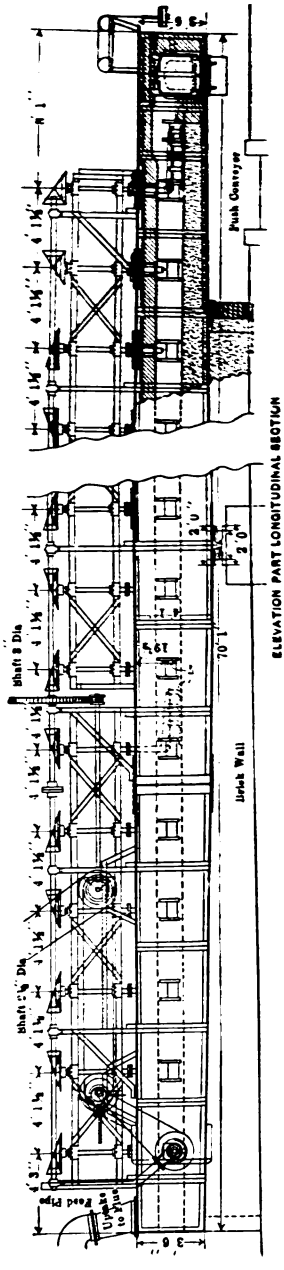


Fig. 38.—Edwards roasting furnace.[7]

The **Merton Furnace** (Fig. 39) is a more compact modification of the Edwards type and consists of three superimposed hearths. Effective action occurs as the ore drops through the flame from one hearth to the hearth next below. The rabbles are similar to those used in the Edwards furnace and are water cooled. This furnace is said to have a greater capacity than the Edwards, and involves smaller initial cost.

The **Ropp Furnace** is of the single-hearth, straight-line type in which a rake operates in place of the usual revolving rabbles.

The **Holthoff-Wethey Furnace** has two hearths, a cooling hearth being arranged below the roasting hearth. The rabbles, consisting of horizontal carriers with a number of blades attached, are driven by chains which are in turn operated by sprocket wheels. The rabble carriers are supported by rollers attached to the chains at each end and operate over the whole width of the hearth.

The **Combination Edwards and Holthoff-Wethey Furnace** (Fig. 40) designed at the Great Boulder Perseverance Mine, Kalgoorlie, utilizes the rabbling system of the Edwards furnace in the top hearth and the Holthoff-Wethey system in the lower hearth.

The **Pearce Turret Furnace** (Fig. 41), now largely superseded by the Edwards, consists of a circular

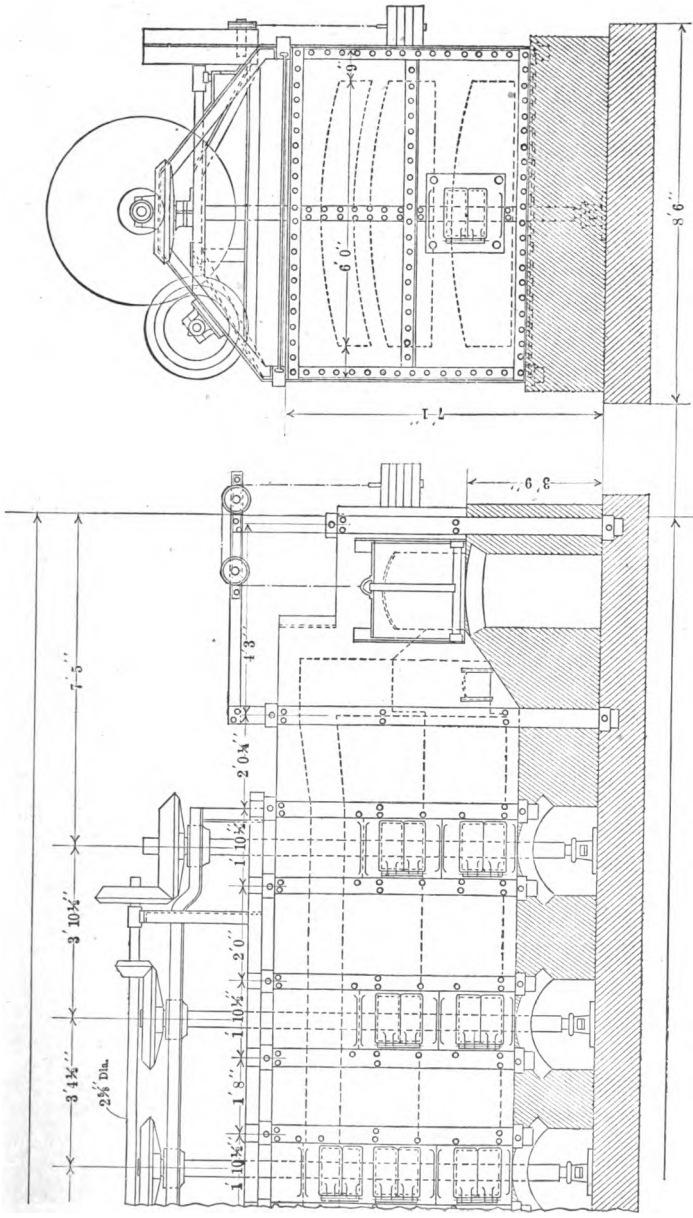


FIG. 39.—Merton roasting furnace (feed end).[7]

reverberatory hearth, single or superimposed, interrupted by a discharge hopper. Rabbling is done by means of rotating water-cooled rabble arms, to which are attached solid plate-steel or bar-iron rabbles.

The **Macdougall Furnace** consists of a number of circular hearths contained in a cylinder or tower. Rabbling is effected by means of a central shaft to which are connected a number of rabbles operating over the ore in each hearth. The ore is rabbled from centre to circumference in one hearth and from circumference to centre in the hearth below. In the former case the ore

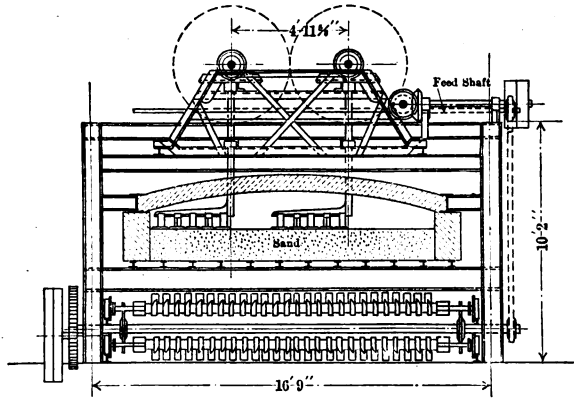


FIG. 40.—Edwards & Holthoff-Wethey furnace.[7]

drops through a pass near the circumference and in the latter case near the centre. The roasting action is materially helped by the direct exposure to the flame during transit from the hearth to the one below. The ore is fed through the roof. Capacity averages about 25 sq. ft. hearth area per ton roasted per 24 hours.

The **Herreshoff Roasting Furnace** (Fig. 42), a modification of the Macdougall roaster, has air-cooled rabbles. The utilization of the heated air in the form of draught has been found to result in an economy in fuel consumption and permits the roasting of a concentrate lower in sulphur content than would otherwise be possible.

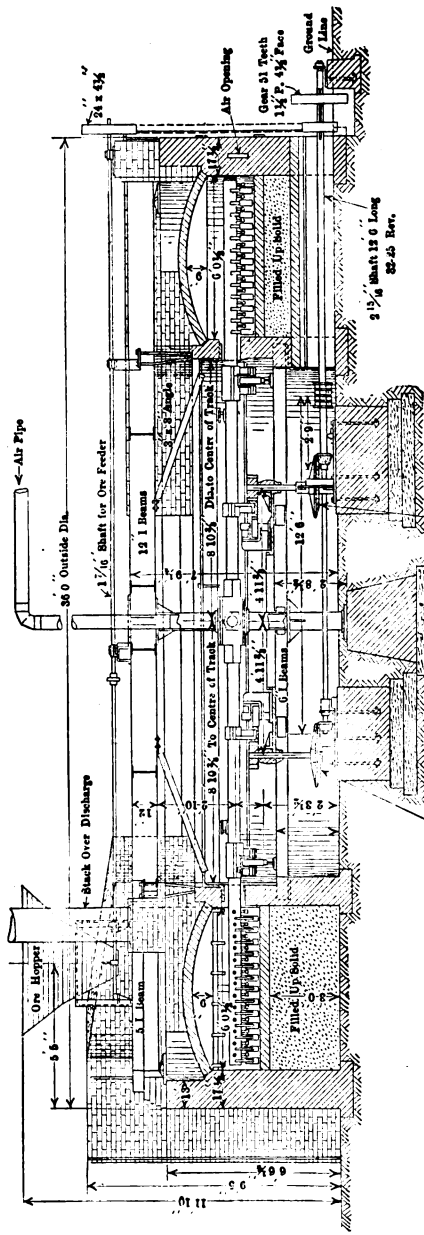


FIG. 41.—Single hearth Pearce turret furnace

In the **O'Brien Furnace** (Fig. 43) the Macdougall principle is followed, but the central driving shaft is hollow and consists of three compartments. The central compartment serves for

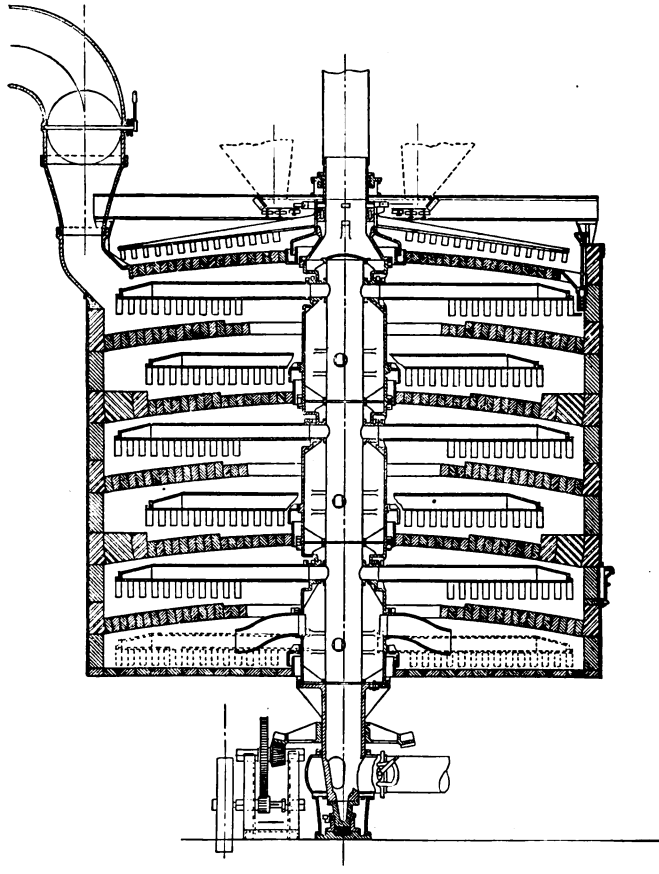


FIG. 42.—Herreshoff roasting furnace.

the passage of cold air to the rabbles and the two others serve as outlets for the hot air. The rabbles are fitted with a division plate placed horizontally. The cold air enters from the shaft and passes under this plate and returns above it to the shaft.

The arms are inserted or removed horizontally, thus insuring space and heat economy and greater capacity.

In the **Wedge Furnace** (Fig. 44) the essential feature is an enlarged central shaft, about 5 ft. in diameter, through which it is possible to remove and replace the rabble arms, and which also facilitates the use of independent air or water connections

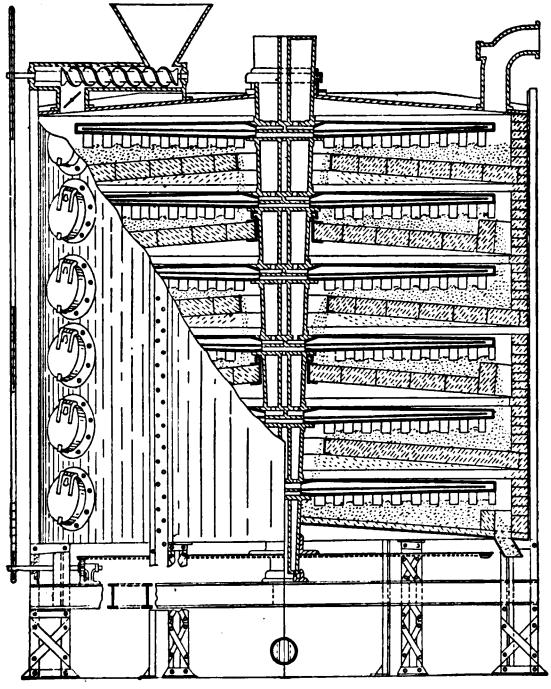


FIG. 43.—O'Brien furnace.

to each arm. Adjustment of practice may be made to suit the requirements of the particular ore being roasted during the operation of the furnace.

In the **Ridge Furnace** (Fig. 45) there are a number of superimposed hearths. The uppermost hearth dries and heats the ore and from there it is fed downward through passes to the

lower hearths, being rabbled en route. The lowest hearth is a cooling hearth where the air required in the furnace is preheated.

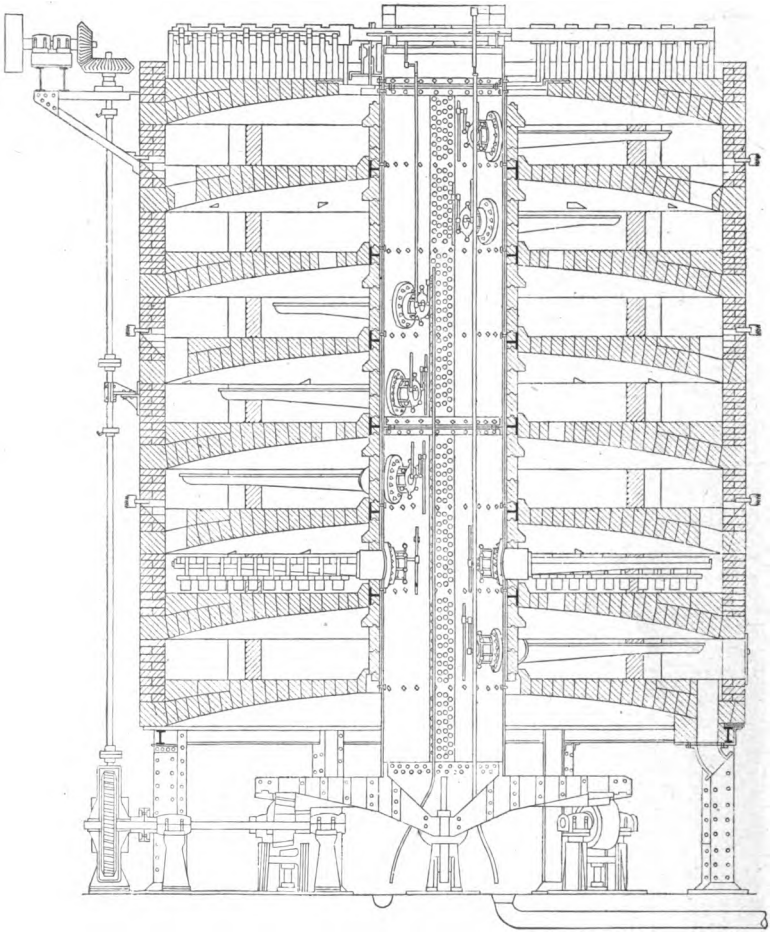


FIG. 44.—Wedge mechanical furnace.

An economy of heat in cooling the ore before it leaves the furnace, and utilizing the heat so recovered, is claimed.

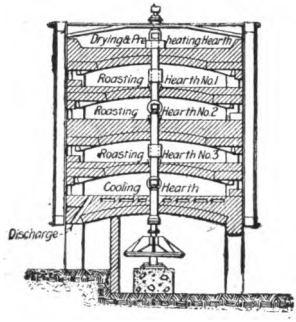


FIG. 45.—Ridge roasting furnace.

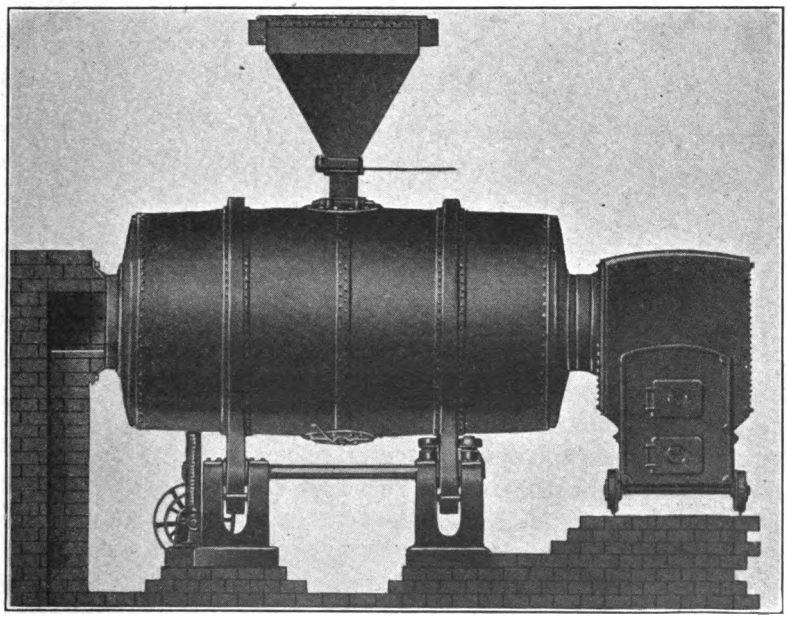


FIG. 46.—Bruckner roasting furnace.

The **Bruckner Roasting Furnace** is of the cylindrical revolving type, and of intermittent operation. The furnace body is supported by double tires and rollers, the latter being operated by means of a shaft driven by worm gear. In the type illustrated (Fig. 46) the fire-box is of the removable type with no fixed connection between it and the furnace.

A **Fine-Ore Feeder** for roasting furnaces is shown in Fig. 47. This consists[7] of a fluted roll operated by a crank and ratchet driven by an eccentric rod off the roasting driving gear. Adjustment is made by altering the throw of the bell crank. The

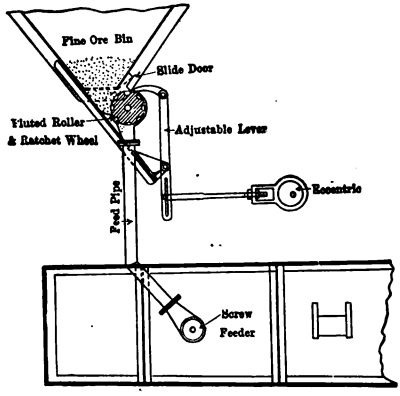


FIG 47.—Fine-ore roasting furnace feeder.[7]

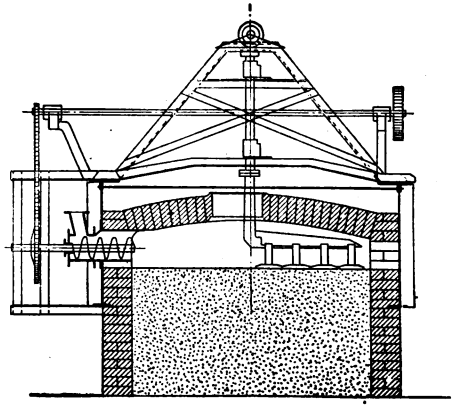


FIG. 48.—Conveyor feeder for furnace.[7]

fluted roll is 11 in. diam. and 14 in. long and has 8 corrugations, each 2 in. in diam. The centres of curvatures of the corrugations are on the circumference of the roll. A cast-iron shell forms the casing to the rolls and is provided with inspection covers to permit the removal of any undesirable material.

Delivery of Ground Ore into Furnace Chamber[7] may be effected by means of a short screw conveyor (see Fig. 48) adjusted to deliver well above the maximum requirements. Actual feed amount is controlled by an adjustable device connected with the ore-feeding arrangements.

SECTION VI

SCREENING

Grizzlies (Fig. 49) are used to classify a coarsely broken or unbroken ore by screening. The screen usually consists of a number of steel bars of tapered section (the thicker edge being toward the ore stream) and spaced according to size and product required. The bars are set at sufficient inclination to permit the ore to traverse the length of the grizzly without need for raking

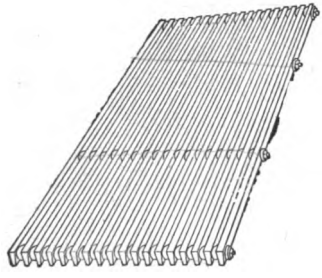


FIG. 49.—Ore grizzly.

or shoveling. They are sometimes placed horizontally in front of the rock breakers and above the ore bins and serve to reduce wear on the shoveling floor. **Shaking Grizzlies** (Fig. 50) have only a limited application and are not widely adopted.

A **Crusher-Feeder** and a **Triple-Screening Grizzly** are shown in Figs. 51 and 52, respectively.

A **Trommel** (Figs. 53 and 54) consists of a cylindrical-, conical-, hexagonal-, or octagonal-shaped screen which is revolved, and is used for washing or classifying ores. Several products of different grades may be obtained by using the trommels in series, by fitting the frame with screen of different-sized apertures, or by con-

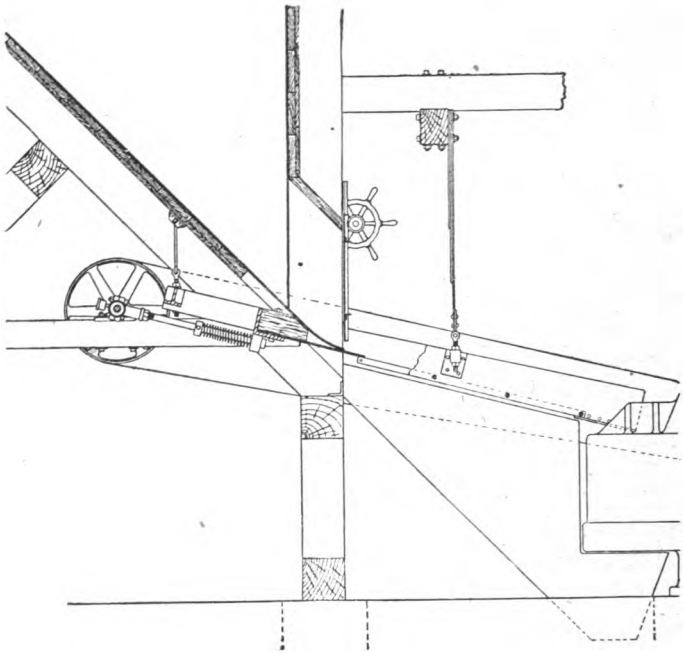


FIG. 50.—Shaking grizzly.[12]

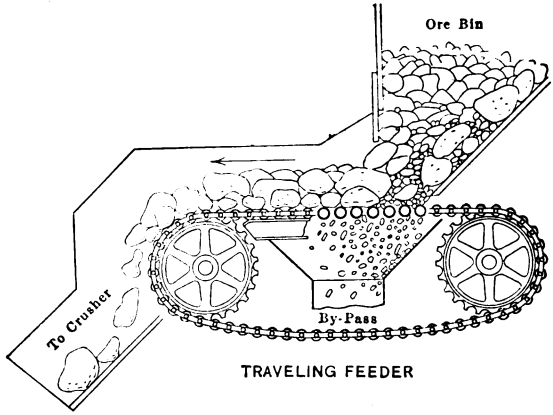


FIG. 51.—Crusher-feeder grizzly.[14]

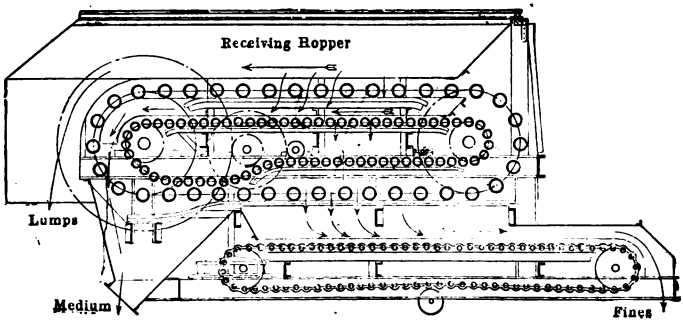


FIG. 52.—Triple screening grizzly.[14]

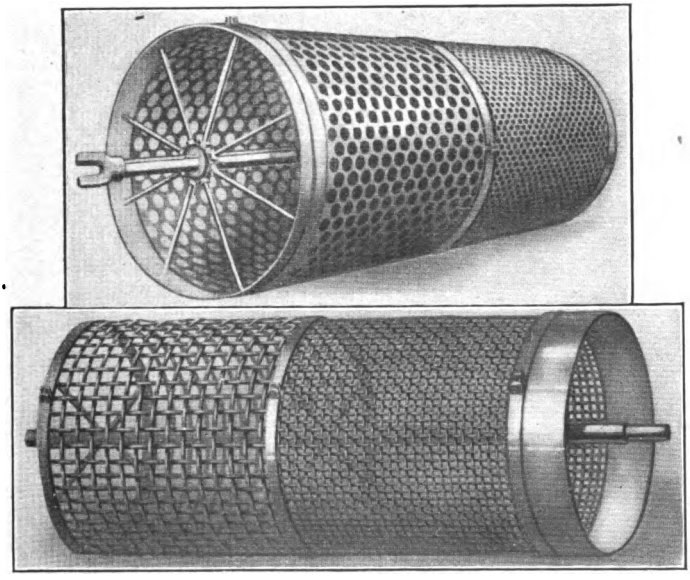


FIG. 53.—Revolving screen.

structing the trommel of several concentric cylinders each of larger diameter and with smaller perforations than the one preceding it.

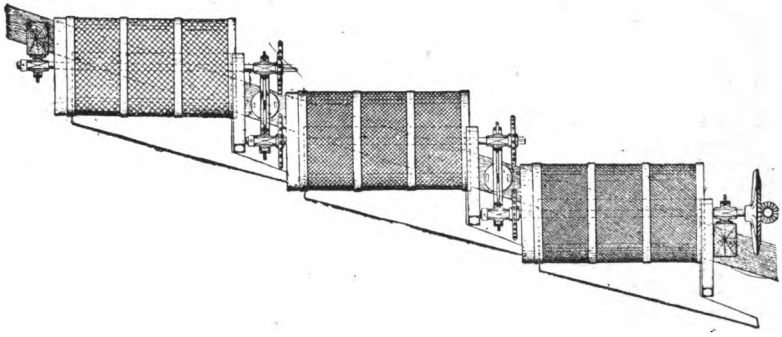


FIG. 54.—Revolving screens operating in series.[19]

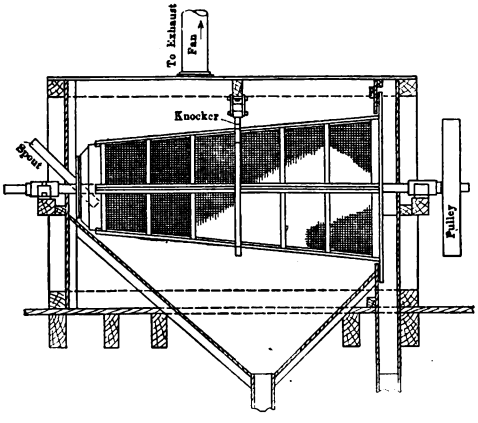


FIG. 55.—Hexagonal revolving screen.[12]

Hexagonal or Octagonal Revolving Screens (Fig. 55) insure better sieving as a result of more energetic impact between material and screen. Screen life is proportionately shorter.

Trommel Perforation sizes, on account of screen-plate inclination and speed, do not coincide with product sizes. A 1-in. cube will require approximately a 1½-in. diameter ring to pass through, and a 1¾-in. opening to be screened by trommel. A statement of comparative measurements for various sizes is given in Table XVI.

The **Maximum Thickness of Steel Plate** through which it is practicable to punch round or square holes of the diameter

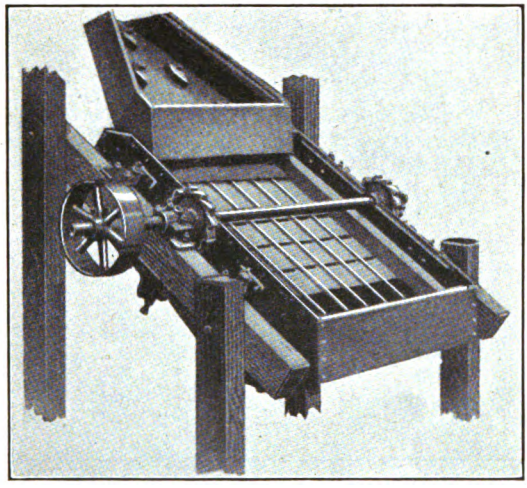


FIG. 56.—Impact screen.[31]

stated is indicated in Table XVII. The figures given are in excess of average requirements.

The **Impact Screen** (Fig. 56) operates by means of a shaft to which two ratchets are attached. The cams force the frame downward against springs, which subsequently carry the frame, on release, outward and upward against the stops. The length of stroke can be regulated, and the impact is such that the entire body of ore is momentarily lifted off the surface of the screen.

TABLE XVI.—SCHEDULE OF RELATIVE SIZES OF STONE AND SCREEN[8]

Size of cube		Size of ring		Size of revolving screen perforation, in.
Exact, in.	Approximate, in.	Exact, in.	Approximate, in.	
0.144	1/8	0.204	3/16	1/4
0.216	7/32	0.306	5/16	3/8
0.289	9/32	0.408	3/8	1/2
0.361	3/8	0.510	1/2	5/8
0.433	7/16	0.613	5/8	3/4
0.505	1/2	0.715	3/4	7/8
0.578	9/16	0.816	13/16	1
0.720	3/4	1.02	1	1 1/4
0.865	7/8	1.23	1 1/4	1 1/2
1.01	1	1.43	1 7/16	1 3/4
1.15	1 1/8	1.63	1 5/8	2
1.30	1 1/4	1.84	1 7/8	2 1/4
1.44	1 1/2	2.04	2	2 1/2
1.73	1 3/4	2.45	2 1/2	3
2.02	2	2.86	2 7/8	3 1/2
2.31	2 1/4	3.26	3 1/4	4
2.60	2 5/8	3.67	3 5/8	4 1/2
2.88	2 7/8	4.08	4	5
3.46	3 1/2	4.90	5	6
4.04	4	5.70	5 3/4	7
4.62	4 5/8	6.50	6 1/2	8
5.20	5 1/4	7.35	7 1/2	9

Impact Screens are said to be preferable to trommels for handling minus ½-in. material, for the following reasons: (1) Higher separation efficiency; (2) lower first cost for equipment and erection; (3) repairs can be made more speedily; (4) greater accessibility to all parts. A statement of approximate capacity for various meshes is given in Table XVIII.

The **Sturtevant-Newaygo Separator** (Fig. 57) operates on the principle of light hammer taps on reinforced screen surface. It has an output grade range from ¼ in. to 180 mesh; and from one to four products can be made with one machine. The capacity of such machines is given in Table XIX.

TABLE XVII.—MINIMUM SIZE HOLES, SQUARE OR ROUND, WHICH MAY BE PUNCHED IN STEEL PLATES[11]

Thickness of plates		Smallest hole	
Birmingham gauge	Inches	Millimeters	Fraction of an inch
26	0.018	3/4	....
24	0.022	1	....
22	0.028	....	3/64
20	0.035	1 1/4	....
18	0.049	1 1/2	1/16
16	0.065	2	5/64
14	0.083	3	1/8
12	0.109	3 1/2	9/64
10	0.134	4 1/2	3/16
8	0.165	6 1/2	1/4
3/16	0.187	7	9/32
1/4	0.25	10	7/16
5/16	0.312	.....	1

TABLE XVIII.—CAPACITY OF IMPACT SCREENS[31]

Size of screen: 4 ft. × 3 ft.	
Mesh	Tons per 24 hr.
4	200–250
10	150–175
16	100–125
20	75– 90
30	50– 60
40	40– 50
60	30– 40
80	20– 30
100	15– 20

The **Bunker Hill Screen** (Fig. 58) is in the form of a cone connected at its base with a hollow shaft placed at an angle of 45°. The screen is driven by gearing, and may be operated at from 20 r.p.m. The feed strikes the screen tangentially, the fines are delivered through the wire cloth, and the oversize

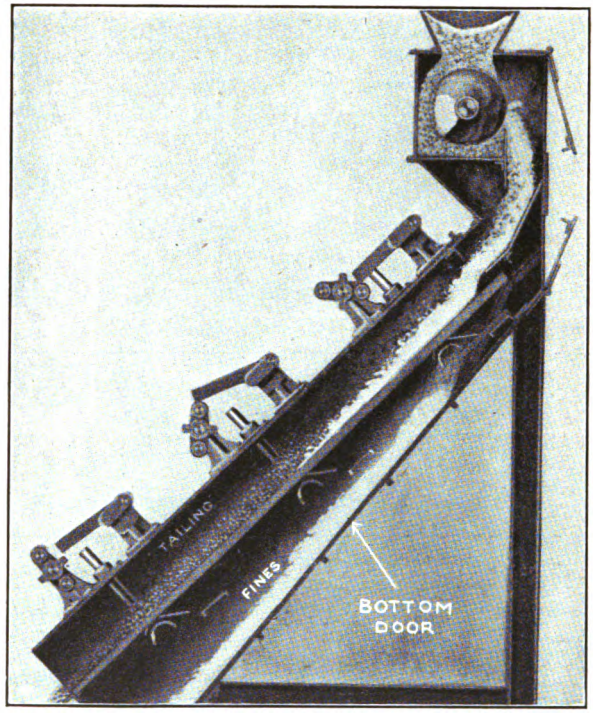


FIG. 57.—Sturtevant-Newaygo separator.[32]

passes through the hollow shaft. A spray of water through the screen from the outside materially assists separation.

The **Callow Screen** (Fig. 59) consists of an endless-belt screen revolving on rollers. The pulp is delivered onto the upper surface of the screen, through which the fines pass. Water sprays are used to clean the screen and assist in the removal of

oversize. Approximate capacities of the 24-in. belt machine are given as follows:

Using 16-mesh screen.....300 tons per 24 hours
Using 60-mesh screen.....150 tons per 24 hours
Using 80-mesh screen.....100 tons per 24 hours.

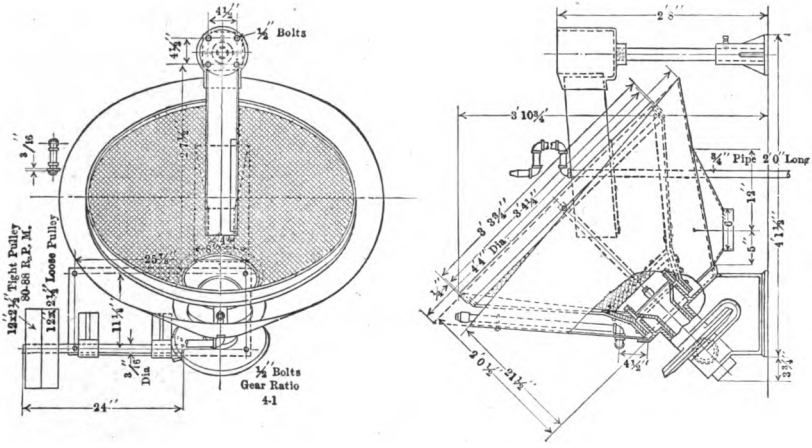


FIG. 58.—Bunker Hill screen.[19]

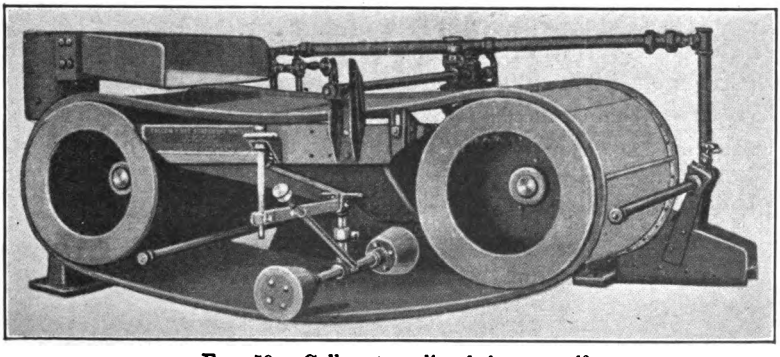


FIG. 59.—Callow traveling-belt screen.[12]

TABLE XIX.—NEWAYGO VIBRATORY-SCREEN OPERATION[32]

Kind of wire	Cloth mesh	Diameter of wire	Opening	Output approx. mesh	Capacity approx. lb. per hour No. 1 separator	Capacity approx. lb. per hour No. 3 separator
Steel	2	0.120	0.380	4	14,000	28,000
	4	0.080	0.170	8	12,000	24,000
	6	0.047	0.120	12	10,500	21,000
	8	0.047	0.078	16	9,000	18,000
	12	0.032	0.051	24	8,500	17,000
	16	0.023	0.0395	30	7,500	15,000
	20	0.018	0.032	40	6,500	13,000
	24	0.015	0.0267	50	5,500	11,000
	30	0.0135	0.0198	60	4,000	8,000
	40	0.011	0.0140	80	2,500	5,000
Brass	30	0.012	0.0213	60	4,000	8,000
	40	0.010	0.015	80	2,500	5,000
	60	0.008	0.0087	100	1,000	2,000
	80	0.00575	0.0068	140	700	1,400
	100	0.0045	0.0055	180	550	1,100

The Number of Holes per Linear Inch in a screen is termed the *mesh.* This figure has no significance unless the wire sizes are also given. The general adoption of a standard series of screens with fixed ratio differences in the mesh apertures is urgently needed.

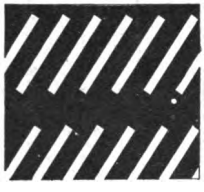
The word **Screening** is used to denote the number of holes per square inch. Thus 40 by 40 mesh is known as 1600 screening. The use of this term to designate screen sizes is to be deprecated, as it conveys no definite information as to the actual dimensions of the screening apertures.

Screen or Mesh Aperture is a term usually referring to the linear dimension of opening from wire to wire. One dimension only is given when the opening is square and two dimensions are given when the opening is oblong.

Needle-Slot Punched Screens are sometimes used for battery and other mill work. The slots are usually about ½ in. in length. Trade sizes of such screening are given in Table XX.

TABLE XX.—NEEDLE-SLOT SCREENS[11]

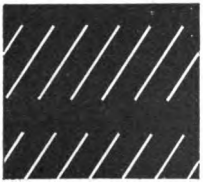
No.	Mesh	Width of slot	U. S. Standard gauge steel	Decimal of an inch	Weight per sq. ft. lb.+
000	6	0.094	16	0.0625	2.55
00	8	0.083	16	0.0625	2.55
0	10	0.070	16	0.0625	2.55
1	12	0.058	16	0.0625	2.55
2	14	0.049	16	0.0625	2.55
3	16	0.042	18	0.05	2.04
4	18	0.035	18	0.05	2.04
5	20	0.029	20	0.0375	1.53
6	25	0.027	20	0.0375	1.53
7	30	0.024	20	0.0375	1.53
8	35	0.022	20	0.0375	1.53
9	40	0.020	22	0.0312	1.275
10	50	0.018	23	0.028	1.147
11	55	0.0165	23	0.028	1.147
12	60	0.015	24	0.025	1 02
13	70	0.0135	24	0.025	1.02
14	80	0.012	26	0.0187	0.765



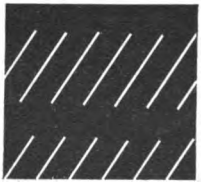
No. 1-(12 Mesh)

No. 3-(16 Mesh)

No. 5-(20 Mesh)



No. 7-(30 Mesh)



No. 10-(50 Mesh)

No. 12-(60 Mesh)

FIG. 60.—Standard diagonal needle-slot screens.[11]

Actual-size diagrams are reproduced in Fig. 60. With this type of screen the total discharge area is small when comparison is made with a wire-mesh, oblong-opening screen of the same aperture width.

Rittinger's Screen Series (Table XXI) is of limited application, the fundamental screen opening being 1 mm. and the ratio between successive openings $\sqrt{2}$.

The **Screen Series** suggested by **Richards,** in which the ratio between successive openings is $\sqrt[4]{2}$, disposes of certain disadvantages of the Rittinger series, but results in an unnecessary multiplication of screens in the coarser sizes. Values of the finer sizes are given in Table XXII.

TABLE XXI.—RITTINGER'S SIEVE SCALE[3]

1.414 = $(\sqrt{2})$ is taken as ratio of diameter of hole in screen to the one above it

Diameters, mm.	Areas if holes are square, sq. mm.	Volumes if particles are cubes, cu. mm.
64.0	4,096.0	262,144.0
45.2	2,048.0	92,668.0
32.0	1,024.0	32,768.0
22.6	512.0	11,583.0
16.0	256.0	4,096.0
11.3	128.0	1,448.0
8.0	64.0	512.0
5.7	32.0	181.0
4.0	16.0	64.0
2.8	8.0	22.6
2.0	4.0	8.0
1.4	2.0	2.8
1.0	1.0	1.0
0.71	0.5	0.35
0.50	0.25	0.125
0.35	0.125	0.044
0.25	0.063	0.016

TABLE XXII.—THE RICHARDS OR DOUBLE RITTINGER RATIO OF SCREEN OPENINGS IN SMALL SIZES[33]

Opening (ratio $\sqrt[4]{2}$ = 1.189)		Mesh	Diameter of wire, in.
Inches	Millimeters		
0.0082	0.208	65	0.0072
0.0069	0.175	80	0.0056
0.0058	0.147	100	0.0042
0.0049	0.124	115	0.0038
0.0041	0.104	150	0.0026
0.0035	0.088	170	0.0024
0.0029	0.074	200	0.0021

TABLE XXIII.—THE TYLER STANDARD SCREEN SCALE[34]

Opening (ratio $\sqrt{2}$ = 1.414)		Mesh	Diameter of wire, in.
Inches	Millimeters		
1.050	26.67	...	0.149
0.742	18.85	...	0.135
0.525	13.33	...	0.105
0.371	9.423	...	0.092
0.263	6.680	3	0.070
0.185	4.699	4	0.065
0.131	3.327	6	0.036
0.093	2.362	8	0.032
0.065	1.651	10	0.035
0.046	1.168	14	0.025
0.0328	0.833	20	0.0172
0.0232	0.589	28	0.0125
0.0164	0.417	35	0.0122
0.0116	0.295	48	0.0092
0.0082	0.208	65	0.0072
0.0058	0.147	100	0.0042
0.0041	0.104	150	0.0026
0.0029	0.074	200	0.0021

The **Tyler Ratio of Screen Openings** uses the Rittinger series from 0.0029-in. opening to 1.05 in. in 18 sieves. Alternate units in the Richards ratio are used for the smaller sizes. Details are given in Table XXIII.

The **Screen Standards Suggested by Stadler** as representing a mathematically correct scale reduction ratio of 1:4 from one inch downward are given in Table XXIV. These are based on a system assuming a regular reduction of the particle to half its volume at each step, giving a ratio of $\frac{1}{\sqrt[3]{2}}$; or, by omitting alternate grades, of $\frac{1}{\sqrt[3]{4}}$.

TABLE XXIV.—STADLER'S SCREEN SCALE[35]

Mathematically correct scale; reduction ratio 1 : 4		Nearest I. M. M. standard laboratory screens		
Mesh aperture		Mesh aperture		
Inches	Millimeters	Number	Inches	Millimeters
1.0	25.4			
0.630	16.0			
0.3969	10.080			
0.250	6.35			
0.1575	4.0003			
0.09922	2.52	5	0.10	2.540
0.06250	1.5874	8	0.062	1.574
0.03937	1.0	12	0.0416	1.056
0.0248	0.630	20	0.025	0.635
0.01562	0.3968	30	0.0166	0.421
0.00984	0.250	50	0.01	0.254
0.00620	0.1575	80	0.0062	0.157
0.00391	0.0992	120	0.0042	0.107
0.00246	0.0625	200	0.0025	0.063

The **U. S. Bureau of Standards Screen Scale** is given in Table XXV and commences with a 200-mesh screen with a square opening of 0.0029 in. and a wire of 0.0021 in. diameter. The ratio increases 1.414 between successive sizes.

TABLE XXV.—U. S. BUREAU OF STANDARDS SCREEN SCALE

Mesh per linear inch	Opening, in.	Opening, mm.	Diameter of wire, in.	Diameter of wire, mm.	Area of openings, sq. in.
...	4.20	106.60	0.375	9.52	17.64
...	2.97	75.39	0.207	5.26	8.82
...	2.10	53.33	0.192	4.88	4.41
...	1.49	37.73	0.149	3.78	2.20
...	1.05	26.67	0.149	3.78	1.10
...	0.742	18.85	0.135	3.43	0.551
...	0.525	13.33	0.105	2.67	0.276
...	0.371	9.423	0.092	2.34	0.138
3	0.263	6.680	0.070	1.78	0.069
4	0.185	4.699	0.065	1.65	0.034
6	0.131	3.327	0.036	0.91	0.017
8	0.093	2.362	0.032	0.81	0.0086
10	0.065	1.651	0.035	0.89	0.0042
14	0.046	1.168	0.025	0.64	0.0021
20	0.0328	0.833	0.0172	0.44	0.00108
28	0.0232	0.589	0.0125	0.32	0.00054
35	0.0164	0.417	0.0122	0.31	0.00027
48	0.0116	0.295	0.0092	0.23	0.000135
65	0.0082	0.208	0.0072	0.18	0.0000672
100	0.0058	0.147	0.0042	0.11	0.0000336
150	0.0041	0.104	0.0026	0.07	0.0000168
200	0.0029	0.074	0.0021	0.05	0.0000084

If the analysis is to be carried finer than 0.0029 in. (200 mesh), the next finer sieve opening in the screen scale series is 0.002 in. (280 mesh).

The **Screen Standards Adopted by the Institution of Mining and Metallurgy** (Table XXVI) are designed so that the diameter of the wire used approximates, as closely as possible, to the width of aperture. The cost of manufacturing the finer mesh screens is considerable, and the system presents no obvious advantages. It lacks a fixed ratio between successive sieve openings.

Screening Analyses are usually made on unit quantities of material, viz.: 100, 200, 500, or 1000 gm., in order to facilitate percentage calculation. The amount to be taken will depend

TABLE XXVI.—I. M. M. STANDARD LABORATORY SCREENS

Number of openings per linear inch	Diameter of wire		Aperture		Screening area, per cent.
	In.	Mm.	In.	Mm.	
5	0.1	2.540	0.1	2.540	25.00
8	0.063	1.600	0.062	1.574	24.60
10	0.05	1.270	0.05	1.270	25.00
12	0.0417	1.059	0.0416	1.056	24.92
16	0.0313	0.795	0.0312	0.792	24.92
20	0.025	0.635	0.025	0.635	25.00
30	0.0167	0.424	0.0166	0.421	24.80
40	0.0125	0.317	0.0125	0.317	25.00
50	0.01	0.254	0.01	0.254	25.00
60	0.0083	0.211	0.0083	0.211	24.80
70	0.0071	0.180	0.0071	0.180	24.70
80	0.0063	0.160	0.0062	0.157	24.60
90	0.0055	0.139	0 0055	0.139	24.50
100	0.005	0.127	0.005	0.127	25.00
120	0.0041	0.104	0.0042	0.107	25.40
150	0.0033	0.084	0.0033	0.084	24.50
200	0.0025	0.063	0.0025	0.063	25.00

on the weights obtained in the different sizings, and it is generally advisable to make a rough preliminary sizing on 100 gm. to see whether the amounts obtained in all sizes are likely to be sufficient to insure accuracy in weighing. Assay requirements will also influence the amount taken when it is necessary to determine subsequently the metallic contents of the various gradings.

The material is usually thoroughly **Dried and Mixed** before the final sample is taken for sieving. In the case where the pulp carries a percentage of comparatively coarse (*e.g.*, ¼ in.) gravel, it is often impracticable to select a final sample for sieving carrying a true percentage of the coarser material. In the latter case it will often be found advisable to mix and weigh the ore while still damp so that no segregation of coarse particles takes place. The sample taken is then dried with care and any loss avoided. The dry weight is recorded and the entire sample screened. The

percentage through each sieve is then calculated from the weights obtained.

The Mixed Method for Grading Analysis is recommended by Stadler and may be described as follows: A convenient weight of the dried sample is transferred to a suitable metal frame having the finest screen it is proposed to use fitted into the bottom. The finest material is then removed by wet screening and the remaining washed product is dried in the same screen frame, and the grading continued and finished on the re-dried sand. The percentage of fines removed by wet screening is estimated by difference.

Advantages Claimed for the Mixed Method are as follows:

(*a*) Greater accuracy.

(*b*) Saving in time in spite of the double drying, as with washed sands in a comparatively short time a clearly defined end of the operation of screening is reached.

(*c*) No dusting.

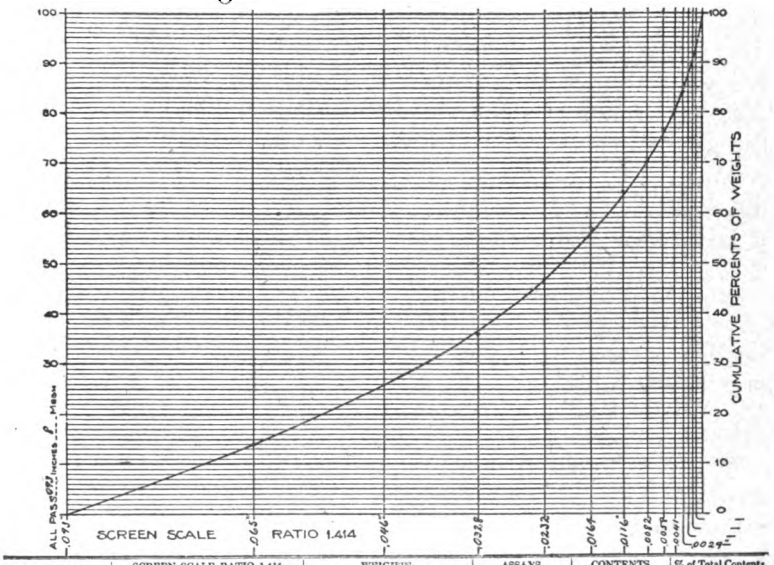
Drying of Samples should be carried out in the final stages on steam or water baths. Stadler has pointed out that the use of a high temperature may result in splitting of the grains and consequent error in grading analysis result.

The **Selection of a Series of Screening Sieves** should be influenced by the fact that all subsequent calculations and graphic presentations of results are simplified when a scale has been adopted in which there is a constant ratio between the openings. In this respect the Tyler series (Table XXIII) is recommended.

Screening Tests are made by placing the weighed portion of the dry ground ore on the mesh of the largest of a series of screens arranged in order, and commencing with an aperture through which the whole of the sample just passes. The nest of screens is then subjected to a rotary shaking motion either by hand or machine. The separation is facilitated by the use, in each screen, of an iron washer, which tends to break up any small balls of slime and to give the screen a certain amount of vibration. Before removing any screen from the one next below it, it is advis-

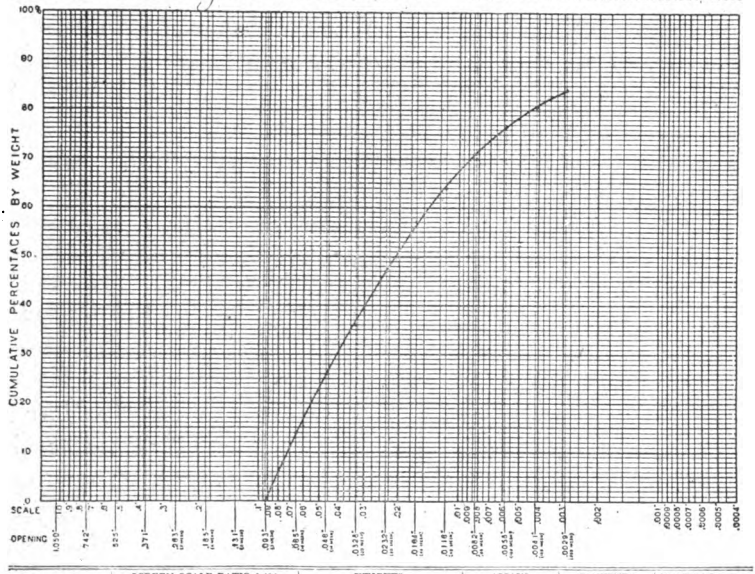
Cumulative Direct Diagram of Screen Analysis on Sample of Roll crushing

Name Midas Gold Mg Co, Red Butte, Idaho. Date November 30, 1912



Indicate the Screen Crushed through and also First Retaining Screen	SCREEN SCALE RATIO 1.414				WEIGHTS			ASSAYS			CONTENTS			% of Total Contents		
	Openings		Mesh	Diameter Wire Inches	Sample Weights	Per Cent	Per Cent Cumulative Weights									
	Inches	Millimeters														
	1.050	26.67		.149												
	.742	18.85		.135												
	.525	13.33		.105												
	.371	9.423		.092												
	.263	6.680	3	.070												
	.185	4.699	4	.065												
	.131	3.327	6	.036												
All pass	.093	2.362	8	.032	Grams											
Retained on	.065	1.651	10	.035	70.0	14.0	14.0									
"	.046	1.168	14	.025	130.0	13.0	26.0									
"	.0328	.833	20	.0172	90.0	18.0	36.0									
"	.0232	.589	28	.0126	55.0	11.0	47.0									
"	.0164	.417	35	.0122	50.0	10.0	57.0									
"	.0116	.295	48	.0092	35.0	7.0	64.0									
"	.0082	.208	65	.0072	30.0	6.0	70.0									
"	.0058	.147	100	.0042	32.5	6.5	76.5									
"	.0041	.104	150	.0026	20.0	4.0	80.5									
"	.0029	.074	200	.0021	17.5	3.5	84.0									
Pass	.0029	.074	200	.0021	80.0	16.0										
				Totals	500.	100.										

FIG. 61.—Graphic statement of screen analysis.[34]



Indicate the Screen Crushed through and also First Retaining Screen	SCREEN SCALE RATIO 1.414				WEIGHTS			ASSAYS			CONTENTS			% of Total Contents		
	Openings		Mesh	Diameter Wire Inches	Sample Weights	Per Cent	Per Cent Cumulative Weights									
	Inches	Millimeters														
	1.050	26.67		.149												
	.742	18.85		.135												
	.525	13.33		.106												
	.371	9.423		.092												
	.263	6.680	3	.070												
	.185	4.699	4	.065												
	.131	3.327	6	.036												
All pass	.093	2.362	8	.033	Grams											
Retained on	.065	1.651	10	.035	70.0	14.0	14.0									
"	.046	1.168	14	.025	60.0	12.0	26.0									
"	.0328	.833	20	.0172	50.0	10.0	36.0									
"	.0232	.589	28	.0125	55.0	11.0	47.0									
"	.0164	.417	35	.0123	50.0	10.0	57.0									
"	.0116	.295	48	.0092	35.0	7.0	64.0									
"	.0082	.208	65	.0072	30.0	6.0	70.0									
"	.0058	.147	100	.0043	32.5	6.5	76.5									
"	.0041	.104	150	.0028	20.0	4.0	80.5									
"	.0029	.074	200	.0021	17.5	3.5	84.0									
Pan	.0029	.074	200	.0021	80.0	16.0										
				Totals	500	100										

FIG. 62.—Graphic statement of screen analysis.[34]

able to test the oversize with the tips of the fingers to see whether the separation has been efficient.

An Inevitable Loss occurs in sieving operations. Where particular care is taken to clean the screens after each test it may be assumed that the loss is in the finest material. The actual weighed proportion of the latter may then be credited with an additional amount (usually varying from one-tenth to one-fifth of 1 per cent. of the original weight of ore), so that the total weight of sieved materials should equal the weight originally taken.

Wet Screening is seldom practised, although it is a useful method where it is found necessary to determine the approximate percentage of fine colloid material (such as clay), as distinct from ground ore, passing the finest screen. The adoption of wet screening results in a greater proportion of fines being accounted for than in dry screening, and it is therefore advisable in reporting screen analyses by wet method to state the fact.

The **Results of Grading Analyses with Fixed-Ratio Sieves may be Plotted** in the manner shown in Figs. 61 and 62. The former is an example of a cumulative direct diagram of the analysis. The vertical lines represent the screen openings. The curve starts at the lower left-hand corner and rises toward the top right-hand corner according to the cumulative percentages of screened products recorded in the analysis. In Fig. 62 the same results are plotted as a cumulative logarithmic diagram in which equal distances on the horizontal scale represent equal ratios, and where the logarithms of the diameters of the openings are used to plot the curve in conjunction with the determined percentages.

SECTION VII

COARSE TO MEDIUM CRUSHING IN JAW, GYRATORY, DISC, OR OTHER MACHINES

In the **Blake-Type Crusher** the movable jaw is supported at the top. This is the usual type of crusher adopted in average-size or small mills. Figure 63 illustrates the design and Table XXVII

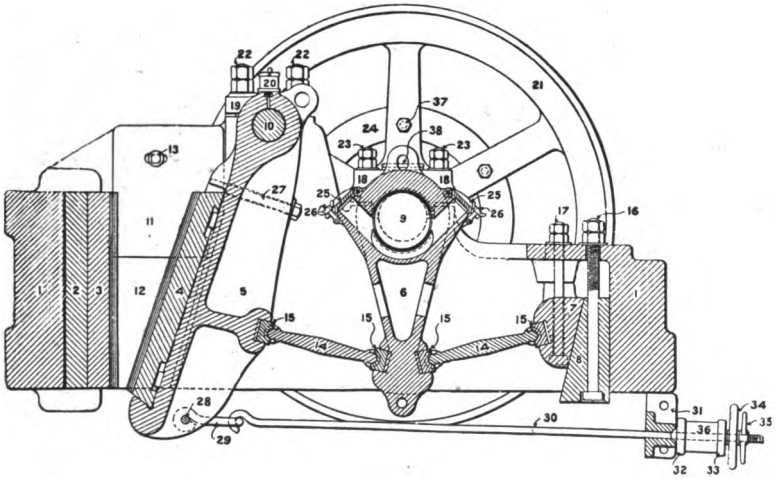


FIG. 63.—Farrel crusher.[36]

1. Main frame.
2. Round back.
3. Fixed jaw plate
4. Swing jaw plate
5. Swing jaw.
6. Pitman.
7. Toggle block.
8. Wedge.
9. Eccentric shaft.
10. Swing-jaw shaft.
11. Upper half cheek plate.
12. Lower half cheek plate.
13. Bolt for cheek plate.
14. Toggle.
15. Toggle bearing.
16. Bolt for wedge.
17. Bolt for toggle block.
18. Cover for main bearing.
19. Cover for swing jaw shaft.
20. Grease cup.
21. Balance wheel.
22. Bolt for swing-jaw shaft cover.
23. Bolt for main bearing.
24. Pulley.
25. Grease-box cover.
26. Bolt and thumb screw.
27. Bolt for swing-jaw plate
28. Shackle pin.
29. Spring-rod shackle.
30. Spring rod.
31. Spring bar.
32. Washer.
33. Washer.
34. Hand wheel.
35. Thumb nut.
36. Rubber spring.
37. Bolt for pulley.
38. Grease-box cover on main bearing.

gives details of operation. Details of a class with unusual feed opening dimensions, designed for fine crushing, are given in Table XXVIII.

The typical **Blake Crusher Diagram,** illustrating the tendency toward invariable composition of product, irrespective of class

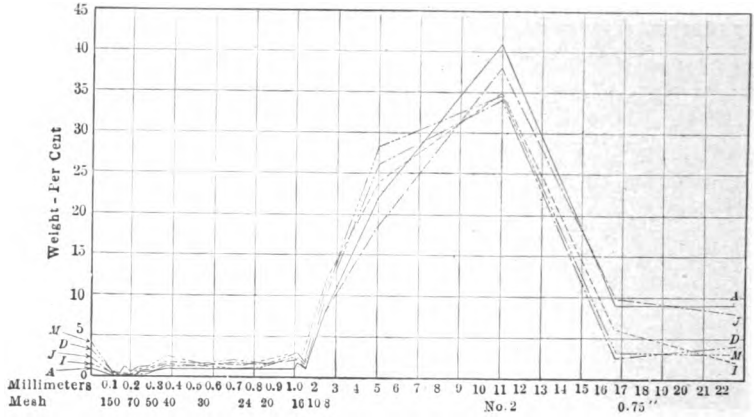


FIG. 64.—Curve showing uniformity of grade produced in crushing different materials in a Blake crusher.[38]

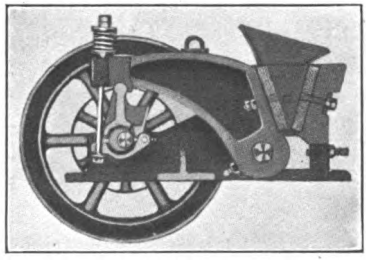


FIG. 65.—Dodge crusher.[39]

of material fed, is shown in Fig. 64. The individual curves refer to different tests with different materials. The crusher jaws were set to 1 inch.

The **Dodge-Type Crusher** (Fig. 65) has its movable jaw pivoted at the lower end and consequently produces a finer and more even product than the Blake. It is not made in large sizes.

TABLE XXVII.—BLAKE CRUSHER DATA[37]

Size receiving capacity in inches	Approximate capacity, in tons, per day of 10 hr. to either one of sizes stated				Size of pulley, in.	Speed	Horse-power,	Approximate total weight, lb.
	Tons In.	Tons In.	Tons In.	Tons In.				
2× 4	.....	.....	.....	.....	.....	.....	.....	163
8×14	100 to 2½	85 to 2	60 to 1½	35 to 1	24×10 (1)	275–300	8–12	6,400
10×16	170 to 3	150 to 2½	125 to 2	100 to 1½	30×12 (1)	275–300	12–18	9,100
12×20	230 to 3	200 to 2½	175 to 2	130 to 1½	36×12 (1)	250–275	18–25	15,000
13×24	300 to 3	260 to 2½	215 to 2	160 to 1½	42× 8 (2)	250–275	20–30	22,900
15×24	400 to 4	350 to 3½	300 to 3	.....	42× 8 (2)	250–275	20–30	22,900
13×30	375 to 3	320 to 2½	275 to 2	200 to 1½	42×10 (2)	250–275	30–40	30,000
15×30	500 to 5	450 to 4	375 to 3	.....	42×10 (2)	250–275	30–40	30,000
18×36	650 to 4	575 to 3½	500 to 3	450 to 2½	42×14 (2)	250–275	50–60	57,000
24×36	1,000 to 6	850 to 5	750 to 4½	700 to 4	48×14 (2)	250–275	50–60	57,000
24×48	1,200 to 6	.....	800 to 4	.....	72×12 (2)	200–225	.....	98,000
30×60	1,500 to 6	.....	1,200 to 4½	.....	90×18 (2)	180–210	.....	120,000

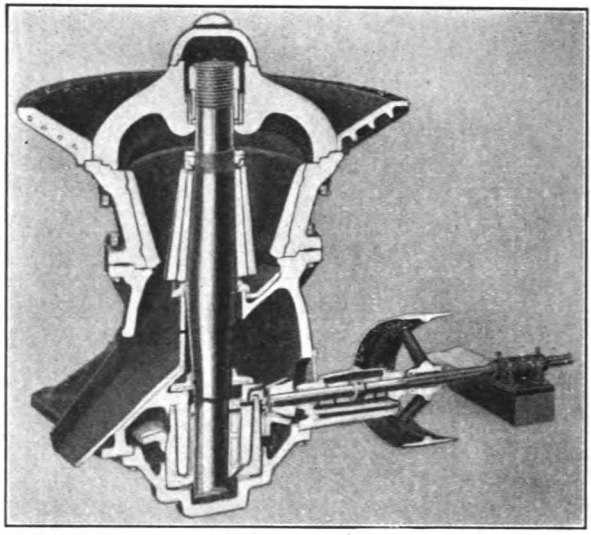


FIG. 66.—Sectional view gyratory breaker.[12]

TABLE XXVIII.—APPROXIMATE CAPACITIES OF FARREL FINE CRUSHERS[36]

Size of receiving capacity	Approximate capacity in tons, per day of 10 hr., to any one of the sizes stated			Size of pulley	Revs. of pulley	Total weight	Weight of heaviest piece
In.	Tons In.	Tons In.	Tons In.	In.		Lb.	Lb.
20 × 4	30 to 1	15 to ½	7 to ¼	48×6½	300	7,000	3,500
30 × 4	50 to 1	25 to ½	12 to ¼	48×11	300	10,500	5,500

TABLE XXIX.—APPROXIMATE CAPACITY OF DODGE CRUSHERS[11]

Size of jaw opening, in.	Horsepower required	No. of tons per hour, nut size	Revolutions per minute
4 × 6	3	½	300
7 × 9	6	1½ to 2½	300
7 × 9 Sectional	6	1½ to 2½	300
8 × 12	10	3 to 5	300
11 × 15	15	6 to 8	250

TABLE XXX.—GYRATORY BREAKER DATA[18]

Size of crusher, No.	Size of each feed opening, in.	Finest setting		Coarsest setting		Size of driving pulley, in.	Revolutions per minute	Horse power required	Approx. weight of crusher, lb.
		Smallest size of product, in.	Capacity in tons 2000 lb. per hour	Largest size of product, in.	Capacity in tons 2000 lb. per hour				
1	5× 20	⅞	4½	1⅞	8½	24× 6	475	4– 6	6,800
2	6× 25	1	6½	2¼	12½	24× 8	450	6– 10	10,000
3	7× 28	1¼	11	2¾	25	28×10	425	10– 15	16,000
4	8× 34	1½	20	3½	48	32×12	400	12– 20	23,000
5	10× 40	1¾	30	4¼	75	36×14	375	20– 25	33,000
6	12× 44	2	50	4½	120	40×16	350	25– 40	45,500
7½	15× 55	2½	80	5	180	44×18	350	45– 70	69,500
8	18× 68	2¾	110	5½	250	48×20	350	65–100	100,000
9	21× 76	3	160	6	350	56×20	325	100–140	158,000
10	24× 84	3½	210	6½	450	56×24	325	115–160	180,000
11	27× 92	4	260	7	550	56×24	325	130–180	195,000
.....	36×130	5	600	8	1,100	72×31	280	200–250	400,000
.....	42×136	5½	700	9	1,300	72×33	280	225–280	420,000

Details of output and other information are given in Table XXIX.

Gyratory Crushers, as their name implies, operate on an eccentric revolving movement of an inverted cone spindle at an adjusted distance from a fixed head. Figure 66 illustrates the type, and Table XXX gives details of construction and operation.

The **Percentage of Various Sized Products from Rock Crushers** may be estimated by reference to Fig. 67. For example when

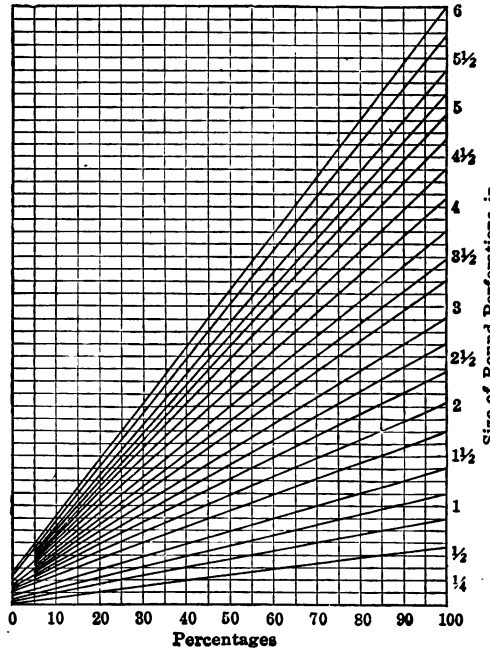


FIG. 67.—Diagram to determine size of rock-breaker product.[8]

crushing to pass a 3½-in. ring 55 per cent. will be found to pass a 2-in. ring. Variation from these figures will occur if the rock breaks in an abnormal manner.

The **Reliance Fine Crusher** (Fig. 68) may be adjusted to produce as fine as ¼-in. product from a coarse feed. The motion to the movable jaw is transmitted through a toggle plate from

an eccentric and roll. Approximate capacities and other information are given in Table XXXI.

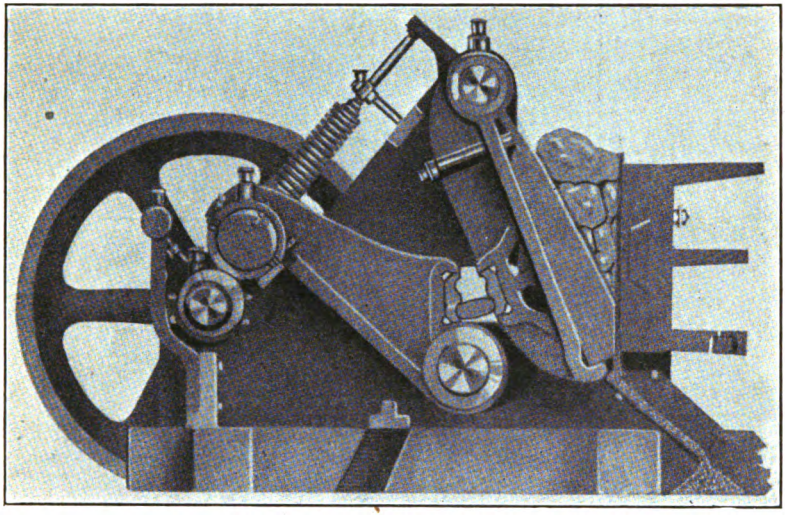


FIG. 68.—Reliance fine crusher.[40]

TABLE XXXI.—RELIANCE FINE CRUSHER DATA[40]

Size of crusher, being the size of opening or receiving capacity of jaws	8″×14″	9″×16″	10″×18″	11″×22″	12″×24″
[1]Capacity per hour in tons to ½″ material and smaller	3–4	4–5	5–6	7–8	9–11
Weight, including foundation timbers, lb	5,200	7,200	10,000	15,000	19,500
Nominal horsepower of engine and boiler required	10–12	12–15	15–20	20–25	30–35
Width of belt	8″	8″ to 10″	10″ to 12″	12″ to 14″	16″
Diameter of belt pulley	28″	30″	32″	36″	38″
Nominal revolutions per minute	325	310	300	290	280

[1] When crushing to ¼″ material at one operation the capacity will be reduced about 1-3 from the figures given in the table.

Comparative Features of Gyratory and Jaw Crushers have been discussed by H. L. Wollenberg, who has summarized his conclusions in condensed form (see Table XXXII). It is stated

TABLE XXXII.—COMPARATIVE FEATURES OF JAW AND GYRATORY CRUSHERS[41]

Crushing from	7 to 1½ in.		10 to 1¾ in.		12 to 2½ in.		18 to 3½ in.		24 to 5 in.		36 to 12 in.		42 to 16 in.	
Type of crusher	J	G	J	G	J	G	J	G	J	G	J	G	J	G
Size of feed opening, in...	10×7	7×56	20×10	10×80	24×12	12×88	36×18	18×136	36×24	24×198	42×36	36×282	60×42	42×345
Capacity in tons per hour	4	7	12	30	25	70	60	130	85	400	160	900	300	1200
Weight in tons.....	4	8	9	18	15	24	29	50	30	95	55	200	100	300
Hourly capacity ÷ tons weight.....	1	0.87	1.25	1.66	1.67	2.91	2.07	2.60	2.83	4.22	2.90	4.5	3	4
Installed h.p.....	8	12	20	25	30	50	65	90	65	150	100	200	150	250
Hourly capacity ÷ installed h.p.....	0.5	0.58	0.60	1.20	0.83	1.40	0.92	1.44	1.31	2.67	1.60	4.5	2	4.8
Height, ft. (disch. spout to feed floor).....	3	5½	3	7½	5	8	6	10	6	15	7	23	8	26
Width, ft. (including the pulleys).....	6½	10	7	13	8	13	13	13	13	20	13	27	15	30
Length, ft. (over all)...	5½	,5	9	6	10	6	12	12	12	12	15	18	16	21

that the gyratory is a machine of greater weight, capacity, and h.p. consumption than the Blake, for the same sized feed and product. Continuity of action is claimed as the reason for

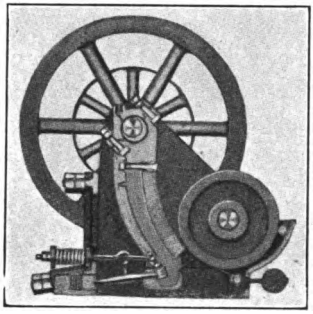


FIG. 69.—Roll-jaw crusher.

economy of power with the gyratory. Floor space is about the same for either type, but there is an economy in height in favor

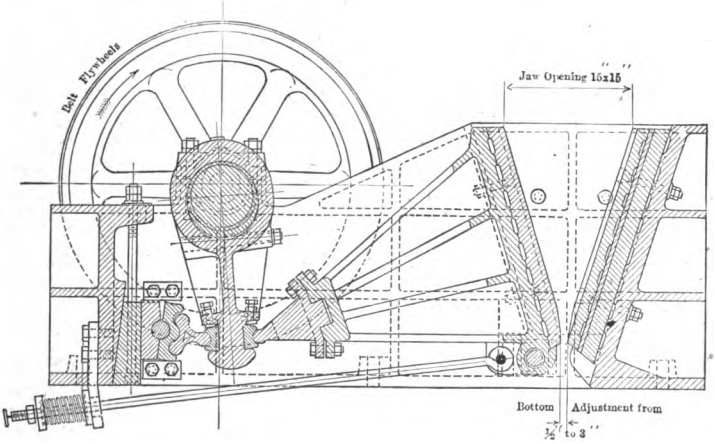


FIG. 70.—Marathon crusher.[42]

of the Blake. The gyratory product is more evenly cubical than that of the Blake. Wear is less on jaw-crusher plates than on gyratory concaves; and inability to reverse liners in the latter

type is a disadvantage which results in higher renewal costs. Repairs are easier made and are less frequent with the Blake than with the gyratory crusher.

Advantages of Gyratory over Jaw Crushers lie in lower first cost, greater automaticity (sometimes eliminating necessity for feed control) and more uniform product. The disadvantages are seen in a comparatively narrow feed opening and more complicated construction. For coarse crushing and very large output the gyratory machine has been widely adopted.

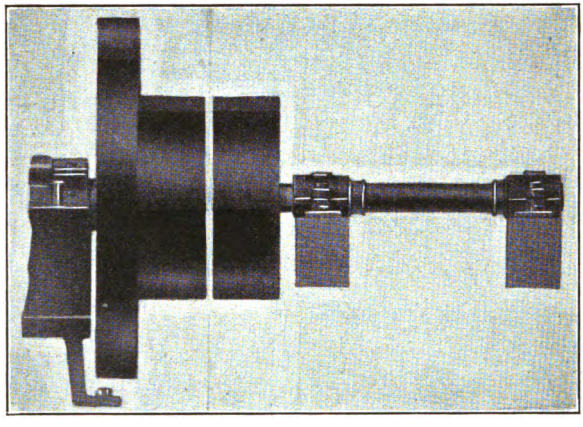


FIG. 71.—Idle pulley arrangements for crusher.

A **Roll-Jaw Crusher,** designed especially for handling sticky or easily caked material, is illustrated in Fig. 69, from which the operation of this type of machine may be understood.

The **Marathon Crusher** (Fig. 70) operates with a triangular toggle. The motions of crushing and freeing the jaw faces reciprocate in both the upper and lower portions of the movable jaw. The latter is supported by shaft and bearings.

Crusher Idler Pulleys in small plants, where several duties are sometimes allocated to one man, are usually a source of trouble as a result of insufficient attention to lubrication when the machine is out of action. To obviate the attention needed in this connection the arrangement shown in Fig. 71 is suitable.

The loose pulley is keyed to an independent shaft and supported on ordinary bearings. There is no connection between

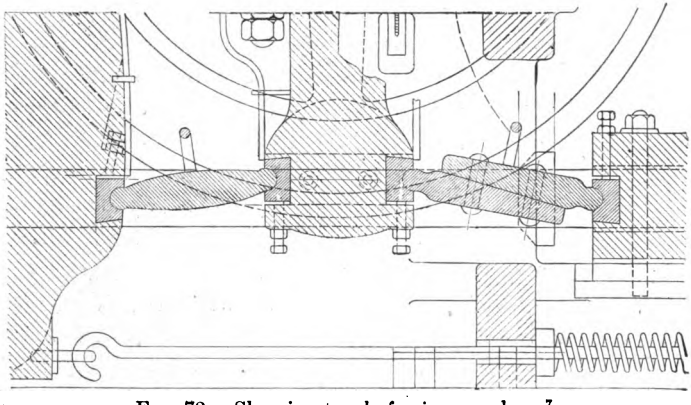


FIG. 72.—Shearing toggle for jaw crushers.[7]

the two pulleys, and no difficulty in "striking" the belt from one to the other.

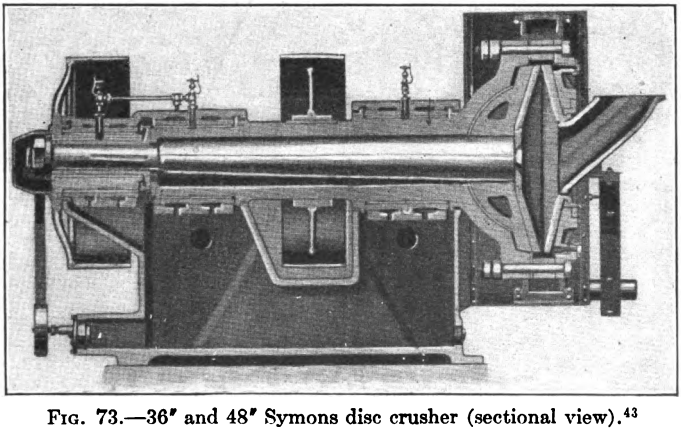


FIG. 73.—36″ and 48″ Symons disc crusher (sectional view).[43]

An effective **Method of Preventing Damage** to jaw-crusher plant and attendant machinery when steel or other unbreakable material finds its way between the jaws is shown in Fig. 72. The

rear toggle consists of two separate pieces, which are lap-jointed in the manner shown, and riveted together so that any abnormal strain put upon them will result in the shearing of the rivets. In the case of gyratory crushers a flange coupling connected with two or more soft-iron bolts in the driving countershaft will serve the same purpose. These methods have been largely adopted in Western Australia.

In the **Symons Disc Crusher** (Fig. 73) the comminution of the ore takes place in the double concavity formed by two hollowed manganese-steel discs. These discs rotate in the same direction and at the same speed, and are supported at an angle to each other. The material is fed centrally between the discs, and is thrown by centrifugal force toward the periphery, where it is crushed. Discharge of ground material takes place in the same direction and is materially aided by the centrifugal action. The ore as it is crushed travels toward an increasing discharge area, thus obviating the lowering of efficiency due to chokage often occurring in other crushing machines. Details of construction, operation, and approximate capacity are given in Table XXXIII.

TABLE XXXIII.—SYMONS DISC CRUSHER DATA[43]

Size of crusher	18″	24″	36″	48″
Shipping weight	5,600 lb.	8,500 lb.	23,500 lb.	39,000 lb.
Size of feed opening	1½″	2½″	3½″	6½″
Minimum exit opening and tons per hour for best results	⅝″, 5– 8	½″, 12–15	¾″, 25–30	1″, 45– 60
Minimum capacity in tons per hour when crushing to size given	½″, 8–10 ¾″, 10–12 1″, 12–15	¾″, 18–20 1″, 20–25 1½″, 25–30	1″, 30–45 1½″, 45–60 2″, 50–60	1½″, 60– 80 2″, 80–100 2½″, 100–120
R.p.m. main pulley	200	200	133	100
R.p.m. eccentric pulley	450	400	300	250
Size of pulleys, in	28×8	34×10	44×14	54×16
Horse power required	12 to 18	18 to 25	30 to 40	50 to 65

Swing-Hammer Pulverizers are high-speed machines and have only a limited application in ore-dressing operations. Various types are constructed in which the action of impact between hammer and rock occurs either on the up or down stroke of the

hammer. Figure 74 illustrates a typical machine of the latter type, but it is only suitable for the reduction of a friable ore.

Hammer-Pulverizer Operation on limestone indicates that the feed should be of a diameter not exceeding 3 in. Softer material

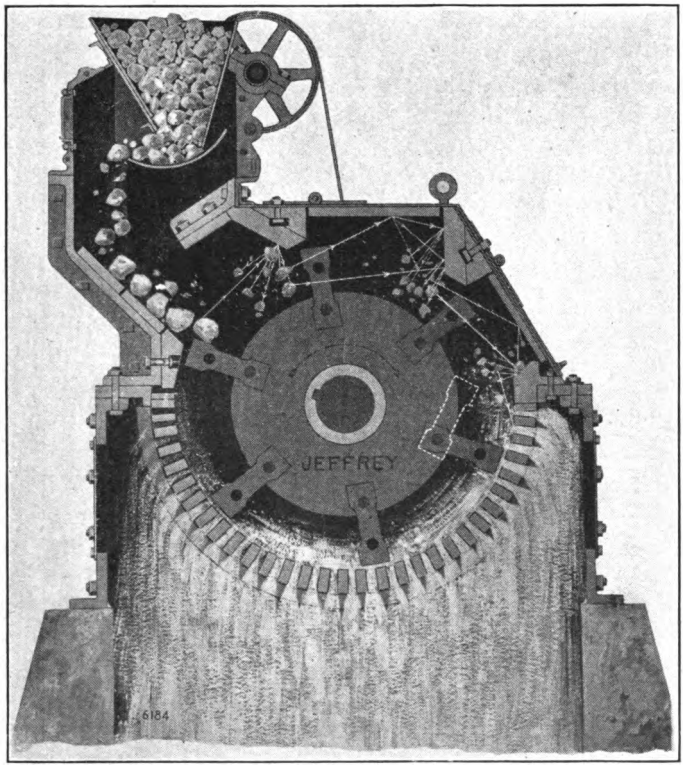


FIG. 74.—Jeffrey swing-hammer pulverizer.[44]

can be successfully handled by this machine in larger lumps. A type of semi-circular grizzly is used to regulate discharge, and the openings may vary from $\frac{1}{8}$ in. to 1 in. A statement of comparative capacities for Jeffrey pulverizers on limestone and coal is given in Table XXXIV.

TABLE XXXIV.—JEFFREY SWING-HAMMER PULVERIZER OPERATION[44]

Limestone

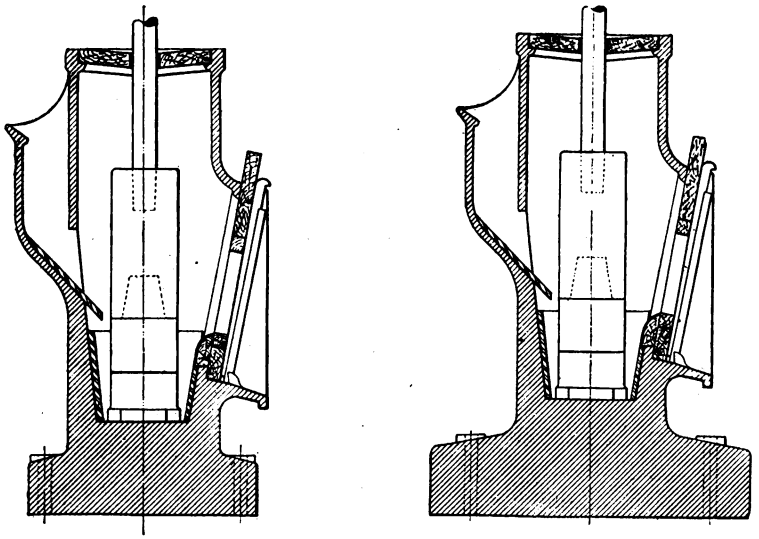
Diameter, in.	Width, in.	Approx. h.p.	Approx. weight, lb.	Speed	Approximate capacity per hour
24	12	15	2,200	1,200 to 1,500	1 to 2 tons limestone, etc., ⅛″ openings.
24	18	25	2,700	1,200 to 1,400	2 to 3 tons limestone, etc., ⅛″ openings.
30	24	40	4,500	1,100 to 1,300	4 to 5 tons limestone, etc., ⅛″ openings.
36	24	60	6,400	1,000 to 1,200	8 to 10 tons limestone, etc., ⅛″ openings.
36	30	75	7,500	1,000 to 1,100	12 to 15 tons limestone, etc., ⅛″ openings.
42	24	75	8,500	900 to 1,000	12 to 15 tons limestone, etc., ⅛″ openings.
42	36	100	9,500	900 to 1,000	15 to 20 tons limestone, etc., ⅛″ openings.
				Coal	
30	24	35	4,300	1,100 to 1,200	20 to 25 tons run mine coal, ¼″ to dust.
36	30	70	7,300	1,000 to 1,100	50 to 60 tons run mine coal, ¼″ to dust.
42	24	70	8,000	900 to 1,000	50 to 60 tons run mine coal, ¼″ to dust.
42	36	80	9,000	900 to 1,000	70 to 75 tons run mine coal, ¼″ to dust.
42	48	110	16,000	800 to 900	90 to 100 tons run mine coal, ¼″ to dust.

SECTION VIII

STAMP MILLING

Stamp Mills are made in a wide variety of types. The number of stamps operating in a single mortar box may be either one, two, three, or five. The latter number forms the usual unit.

Mortar Boxes set on concrete foundations require a wider base than for a wooden foundation. Types are illustrated in



For wooden foundation or iron sub-base. For concrete foundation.

FIG. 75.—Mortar boxes.[16]

Fig. 75. Single-discharge boxes are in common use, the double-discharge type (back and front), having been generally discarded whenever adopted. Open-front boxes permit greater freedom in changing shoes and dies, and in carrying out general

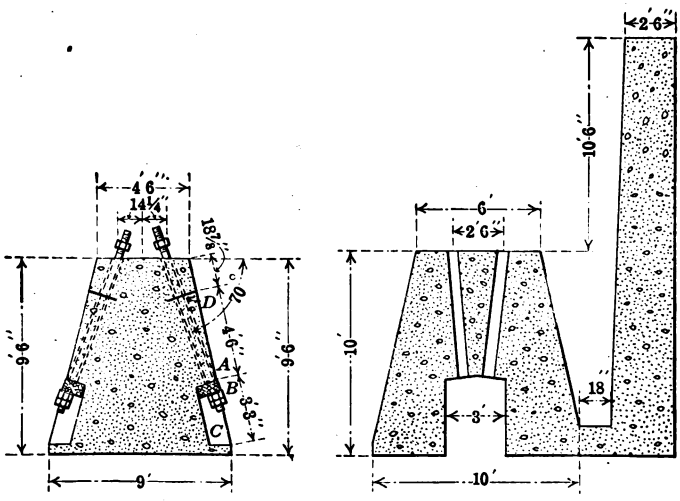
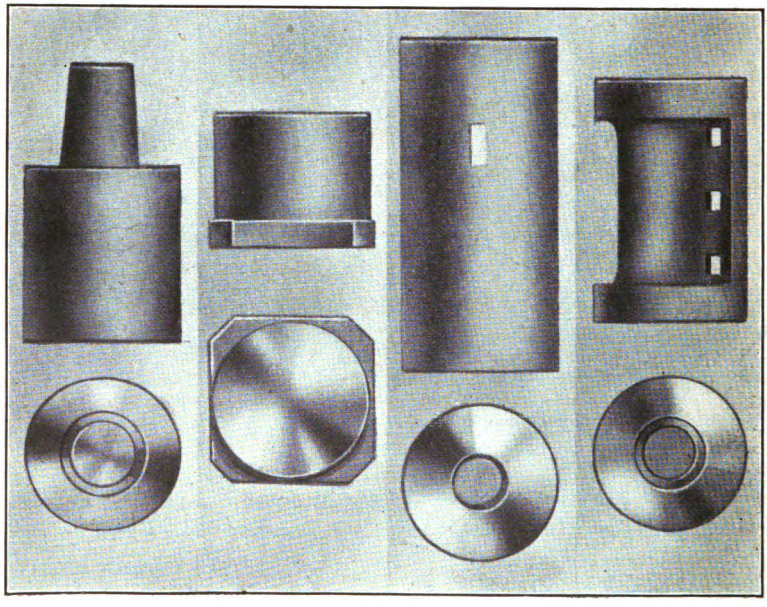


FIG. 76.—Types of mortar foundations.[45]



Shoe Die Bosshead Tappet

FIG. 77.—Stamp-mill parts.

repair work, and were introduced to facilitate the removal of the bulky bosshead used in heavy-stamp mills.

Concrete has now generally superseded wood for mortar-box foundations. Its rigidity, low first cost, neat appearance, and durability are in its favor. Concrete mortar blocks require to be massive, with a wide base, and with accessible bolt holes. Types shown in Fig. 76 are commonly adopted.

Stamp-Mill Parts are illustrated in Figs. 77 and 78. **Shoes** and **Dies** are made of a variety of materials, from mixtures of

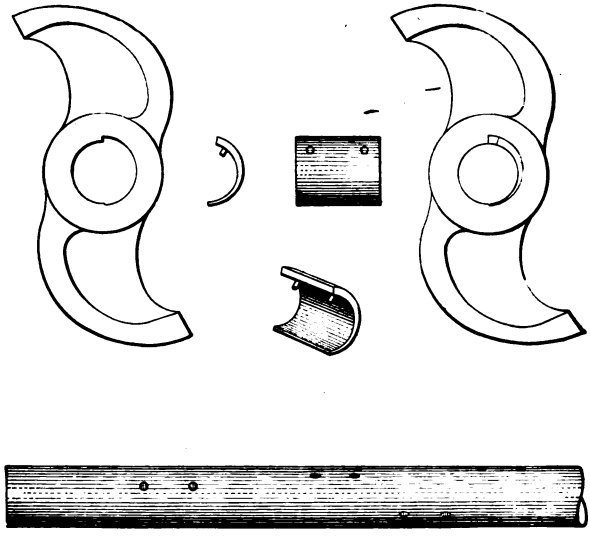


FIG. 78.—The Blanton cam.[11]

scrap iron and steel, cast locally on the property, to the hardest chrome steel. Local conditions and economic considerations should decide the matter of suitability of material used. **Tappets** and **Cams** are usually made of cast steel, **Camshafts** of hammered iron, and **Stems** of the best refined mild steel. **Mortar-Box Liners** are preferably made of the hardest material possible, and manganese steel is usually adopted. A section of a stamp

battery showing the position of the various parts is given in Fig. 79.

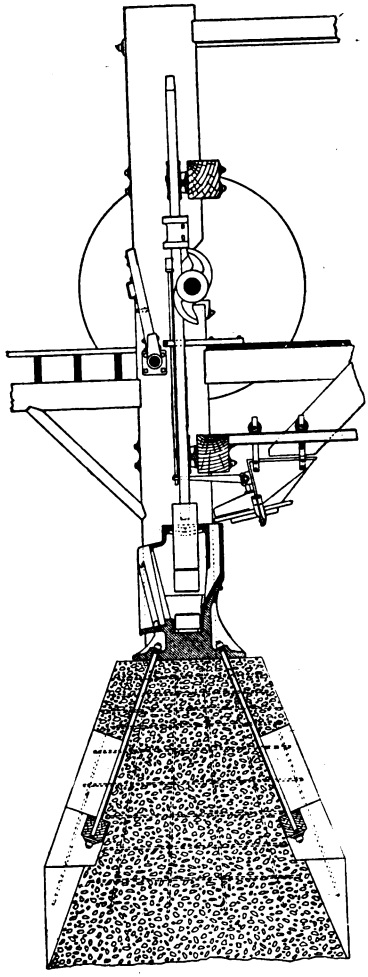


FIG. 79.—Details of stamp-mill on concrete foundation.[12]

Cam Lifting Surface is usually in the form of an involute. In the ordinary design the curve is formed by selecting a point on the outer circumference of a circle representing the hub periphery, and with a radius of length *ab*, about 1.2 times the diameter of the circle, the involute is formed as shown in Fig. 80.

Heavy Stamps require a modification of cam involute design. Fig. 81 shows the Behr type which is used in South Africa. A considerable portion of the cam surface is circular in form, the remainder being of the ordinary involute design. An increased drop-height is obtained with this type, with an evener curve surface.

The **Blanton Self-Tightening Cam** (Fig. 78) relies on a brass taper bushing carrying studs. The latter pocket into holes bored in the camshaft and merely serve to keep the wedge in position. There is no shearing strain on these pins. Cam fastenings of this or a similar type are universally used in stamp mills.

Compensating Weights (Fig. 82) to allow for decrease of effective stamp weight as a result

of shoe wear are usually made to clamp on the stem just above the bosshead. Dimensions are governed by stem and bosshead sizes, and weight required.

Individual False Dies are used to raise the die, when worn down, to the proper level below the screen discharge. These

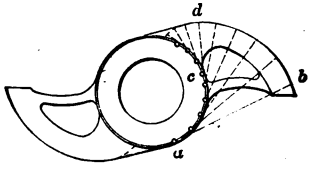


FIG. 80.—Cam involute design.

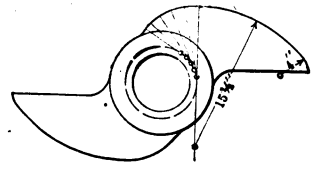


FIG. 81.—Behr-cam involute design.[46]

may be cast of any desirable thickness, of a similar size to the die bottom, and are preferred to the single false die sometimes supplied with mortar boxes. Old dies, worn to an even face, are sometimes used.

The **Height of Discharge** measured from the highest die level to the lowest screen level is regulated, when necessary, by the use of **Chuck Blocks.** These are made of wood of the same length and thickness as the screen frame, and faced inside with steel plate to withstand the scouring action of the pulp.

Battery Framing is usually of the back-knee type. The king-posts are tied to the ore bins by means of a wooden stay held in position either by long bolts or attached by means of angle-plates at each end. The countershaft driving the mills is placed on the streak sills behind the battery. Figure 83 illustrates the usual arrangement of battery timbers and the method of driving.

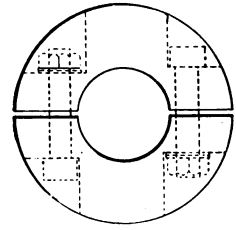


FIG. 82.—Top view of stamp-mill compensating weight.

The principal battery timbers, known as **King Posts,** standing at either end of the mortar box, are connected with concrete foundations by means of cast-iron shoes (Fig. 84). The use of metal posts in place of battery timbers has not proved generally

successful. This is usually due to breakage, consequent on vibration where heavy stamps are used.

Camshaft Bearings (Fig. 85) are seldom provided with more than canvas dust covers. Bearing metal is also rarely used. Such bearings are made of cast iron turned to fit the camshaft.

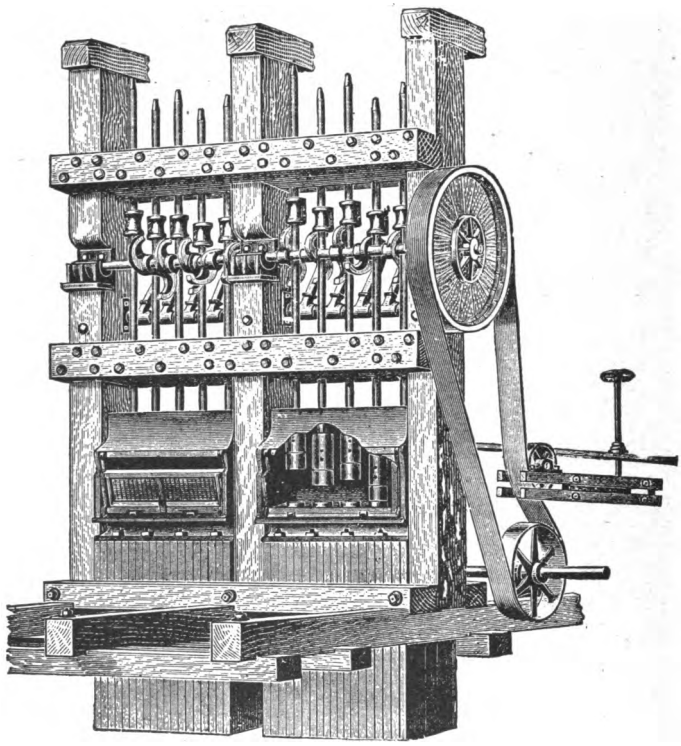


FIG. 83.—Battery framing.[18]

Tappets with one gib and two or three keys are used for light stamps. Stamps whose effective falling weight is beyond 1500 lb. are generally provided with double-gib, three- or four-key tappets.

The faces of **Gibs** used in wedging the tappet on the stem should have a curve of slightly smaller diameter than the stem.

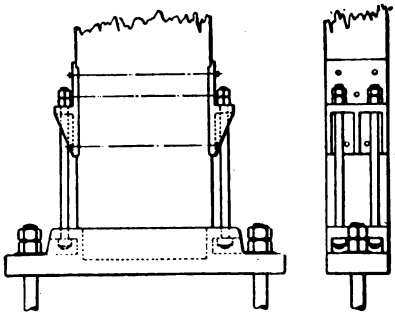


FIG. 84.—Battery-post shoe.[16]

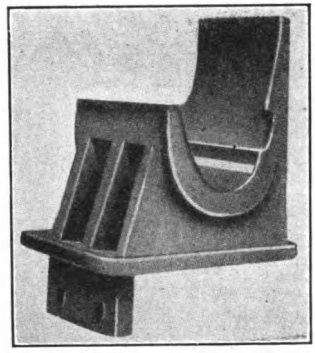


FIG. 85.—Cam-shaft bearing.[16]

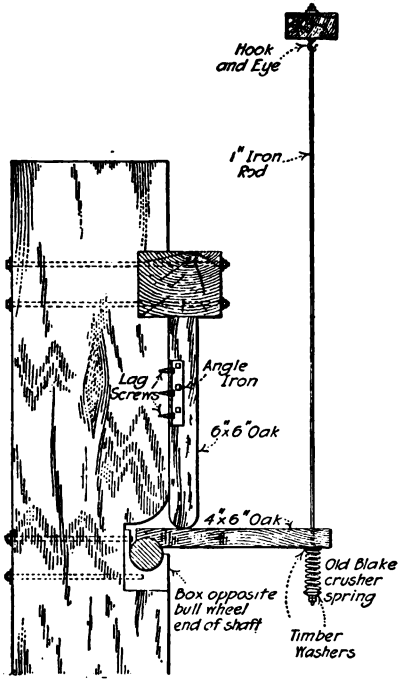


FIG. 86.—Cam-shaft damper.[47]

In addition, it is advisable to channel or counterbore the opposite side of the tappet so as to give additional points of friction contact.

The **Camshaft Damper,** illustrated in Fig. 86, consists of two short pieces of oak, an iron rod, and any suitable spring. The adoption of this device is said to have reduced vibration of the camshaft, crystallization and fracture of the steel, and damage to camshaft bearings.

Stamp-Stem Guides in old mills are often made of wood, but the individual-metal type is now almost universally adopted in new mills. The guide illustrated in Fig. 87 has interlocking bushings in halves, and the guide frames are made with tapered

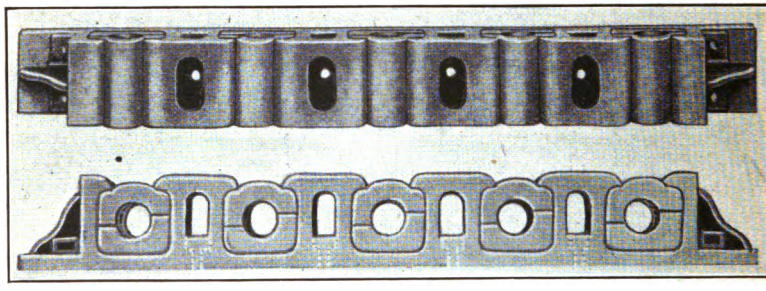


FIG. 87.—Metal stamp-stem guides.[16]

sockets. Stamp stems may be changed without loosening nuts or removing bolts.

A **Finger Jack** is used in conjunction with a **Cam Stick** to "hang up" any particular stamp. The cam stick, generally made of old belting with a wooden handle, is of sufficient thickness when placed between cam and tappet to raise the latter so that the finger jack may be thrown in against the stem to support the tappet above cam action.

Correct **Order of Drop** in a five-stamp mill is a matter of individual opinion among operators. The 1-3-5-2-4 order is largely adopted, but an evener discharge and higher battery output have been claimed in the use of the 1-5-2-4-3 sequence.

The **Nissen Stamp** (Fig. 88) operates in a single mortar box and is especially adapted for high-duty performance on hard or

medium-hard ores. The screening is arranged to encompass the die and extends around it for the greater part of its circumference.

Exhaustive data dealing with the operation of **Heavy Stamp**

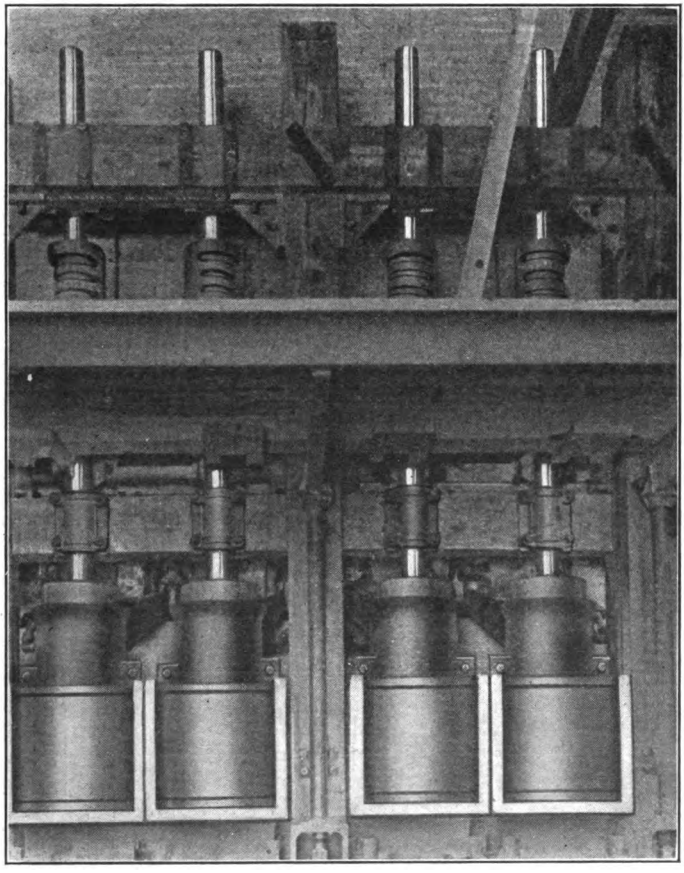


FIG. 88.—A Nissen-stamp mill.

Mills are given in Table XXXV and refer to the results of a series of tests made to compare the work of the **Nissen Stamp** with that of an ordinary five-stamp battery at the City Deep Mine, South Africa, in 1911. Relative mechanical values of the

TABLE XXXV.—COMPARATIVE TESTS BETWEEN NISSEN AND ORDINARY HEAVY GRAVITY FIVE-STAMP MILLS[48]

Test number	1*a*	1*b*	2*a*	2*b*	3*a*	3*b*	4*a*	4*b*	5*a*	5*b*
Number of stamps and kind	4 Nissen	2–5 stamp batteries	4 Nissen	2–5 stamp batteries	4 Nissen	2–5 stamp batteries	4 Nissen	2–5 stamp batteries	4 Nissen	2–5 stamp batteries
Weight of stamps in lb	1932	1863	1927	1859	2245	1855	1993	1775	1991	1773
Number of drops per minute	103	100	103	100	103	100	103	100	103	100
Height of drop, in	8½	8½	8½	8½	8¾	8¾	8½	8½	8½	8½
Height of discharge, in	2⅝	2¼	2	2⅜	2	2½	2¼	2¼	2¼	2¼
Class of screen	Tyler ton-cap		Tyler ton-cap		sq. mesh	14(0.08)	sq. mesh	wire	sq. mesh	wire
Mesh of screen	9	9	9	9	⅜″	⅜″	9	9	⅜″	⅜″
Width of aperture, in	0.205	0.205	0.205	0.205	0.375	0.375	0.277	0.277	0.375	0.375
Length of aperture, in	0.536	0.536	0.536	0.536	0.375	0.375	0.277	0.277	0.375	0.375
Tons dry ore crushed	180.86	192.75	180.25	192.12	193.88	206.66	185.32	194.52	185.7	191.2
Total run in hours	44.33	25.33	39.0	23.166	31.70	21.83	37.29	22.28	29.52	18.85
Tons per stamp per 24 hours	24.47	18.26	27.73	19.90	36.69	22.72	29.81	20.95	37.74	24.34
H.p. per stamp	4.09	4.21	4.09	4.21	4.95	4.21	4.23	4.65	4.23	4.65
H.p.-hours per ton crushed	4.0	5.5	3.56	5.06	3.24	4.45	3.41	5.32	2.71	4.56
Lb. ore crushed per pound falling weight per day	25.33	19.6	28.72	21.412	32.68	24.49	29.91	23.60	37.91	27.45
Water:ore weight	5.4:1	2.7:1	4.2:1	2.4:1	3.1:1	2.15:1	3.3:1	3.3:1	3.03:1	3.26:1
Consumption metal in shoes in pounds per ton crushed	0.129	0.145	0.139	0.24	0.198	0.187	0.147	.....	0.112	0.08

TABLE XXXVI.—DIMENSIONS AND WEIGHTS OF STAMP-BATTERY PARTS[16]

Weight of stamp, lb.	850	1000	1050	1250	1509	1600	1800
Shoe, dia. of body by height, in.	8½×8	9×8	9×8	9×10	9×12	9×12	9×12
Shoe, weight, lb.	150	170	170	200	242	242	242
Die, dia. of body, in.	8¾	9¼	9¼	9¼	9¼	9¼	9¼
Die, 6 in. high, weight, lb.	106	122	122	122	122	122	122
Die, 7 in. high, weight, lb.	121	140	140	140	140	140	140
Die, 8 in. high, weight, lb.	136	157	157	157	157	157	157
Bosshead, dia. by height, in.	8½×18	9×18	9×19	9⅛×22	9¼×26½	9¼×30½	9¼×37
Bosshead, weight, lb.	232	270	285	350	420	498	621
Tappet, 3 keys, dia. by height, in.	9×12	9⅛×14	9⅛×14	9¼×15	9½×15	9½×15	9½×17
Tappet, weight, lb.	136	155	155	170	185	185	212
Cam, dia. of hub, in.	12	12	12	13½	14	14	14
Cam, weight, not less than, lb	180	200	200	225	235	235	240
Stem, dia. by length, in.	3⅛×168	3⅜×168	3½×168	3¾×174	4×186	4×192	4×204
Stem, weight, lb.	365	426	455	545	661	683	725
Camshaft, dia. (length 14½ ft.), in.	5⅜	5¾	6	7	7	7¼	7¼
Camshaft, weight, lb.	1120	1280	1400	1915	1915	2050	2050
Pulley for camshaft, dia. by face, in.	72×14	78×16	84×16	84×18	84×20	84×22	84×24

products were obtained by screening analyses and calculation by means of Stadler's energy-unit system, based on Kick's Law, with the result that the relative efficiency of the Nissen was calculated at 0.25 and that of the ordinary stamp at 0.19, an advantage of 35 per cent. in favor of the Nissen.

Dimensions and **Weights of Stamp-Battery Parts** (U. S. A. practice) may be found in Table XXXVI. Further details

TABLE XXXVII.—WEIGHT AND SIZE OF MILL PARTS FOR VARIOUS WEIGHTS OF STAMPS IN SOUTH AFRICAN MINES

	Simmer & Jack Prop.	Simmer East	City Deep
Weight of stamp, lb	1250	1550	2000
Weight of shoe, lb	160	286	290
Head, lb	365	376	872
Stem, lb	580	726	556
Tappet, lb	145	250	282
Size of shoe, in	9 × 9	9¼ × 14	9 × 14
Head, in	9 × 18	9¼ × 24	9½ × 50
Stem, in. × ft	3½ × 18	4 × 17	4 × 13
Tappet, in	9¼ × 13	9⅝ × 13	9 × 20
Diameter of camshaft, in.	6½	6½	7 hollow 1½ bore

TABLE XXXVIII.—STAMP-MILLING DATA OF AMERICAN PRACTICE[49]

	Tons milled per 24 hr.	Number of stamps	Running weight, lb.	Discharge aperture, in.	H.p.-hours per ton milled	Stamping cost per ton milled, cents	Sizing	
							Per cent., − 100	Per cent., − 200
							Inches	
							0.0055	0.0029
Mex. Silver Mill	1000	60	1450	0.371	4.8	10.0	....	14.0
Nipissing	245	40	1450	0.334	12.1	23.8	20.3	16.7
Belmont	500	60	1200	0.131	7.2	20.7	28.3	19.5
Silver Peak	500	120	1050	0.023	13.8	31.4	54.0	40.0
Homestake	4500	1020	850	0.022	11.0	20.4	80.0	60.0
Hollinger	585	60	1400	0.263	8.6	16.0	....	15.0
Porcupine Crown	150	20	1000	0.250	6.0	14.0	....	15.0
Liberty Bell	485	80	750	0.041	7.8	19.1	54.0	46.0

TABLE XXXIX.—CANADIAN AND UNITED STATES STAMP-MILLING PRACTICE[50]

Mill	No. stamps	Weight, lb.	Drop, in.	Drops per min.	Feed character	Av. Size, in.	Screen used	Regrinding machine
Goldfield Con.	100	1050	7	108	Alunite	1½	4 mesh	Chilean mill
Homestake	1000	920	9	88	Var.	...	30–35 mesh	Tube mills
Mexican	10	1250	6	104	Quartz and clay	2	4 mesh	Chilean mill
Dome	40	1250	6½	100	Quartz and schist	1½	10 mesh	Tube mills
West End, Tonopah	20	1200 1300	6½	101	Quartz	1	.....	Tube mills
Montana Tonopah	40	1100	7½	102	Quartz	2	25 mesh	Tube mills
Tonopah Extension	30	1050	7	100	Quartz	2	3 and 12 mesh	Tube mills
McNamara	10	1400	8	98	Quartz	2	8 and 12 mesh	Tube mills
Nevada Hills	20	1250	6½	107	Hard quartz	1½	8 mesh	Tube mills
Nevada Wonder	10	1400	8	96	Quartz	2½	⅝ in.	Chilean and tube mill
North Star	40	1000	7	110	Quartz	2	30-mesh	None
Black Oak	20	1250	6	102	Quartz	2	20 and 6 mesh	Tube mill
Gold Road	40	1050	7	104	Quartz and calcite	2½	4 mesh	Tube mills
Tom Reed	20	1250	6	105	Quartz	3	12 mesh	Tube mills
Vulture	20	1600	6½	98	Quartz	2	40 mesh	Pans

TABLE XL.—STAMP-MILL DATA OF WESTERN AUSTRALIAN PRACTICE[7]

Mine	Weight of stamps, lb.	Wear per ton crushed		Screening	Discharge, in.	Drops		Duty per stamp per 24 hours, tons
		Shoes, oz.	Dies, oz.			No. per min.	Height, in.	
Ivanhoe.....	1192	....	....	15 × 15	2	104	7½	5.5
Oroya Brown Hill...	1100	4.26	4.56	10 × 10	2	108	7¾	6.48
Sons of Gwalia.	1000	....	....	20 × 20	3½	108	8	6.68
Lake View Consols.....	1200	4.3	2.5	300 mesh	2	102	8	5.5
Golden Horseshoe....	1274	2.14	3.04	15 × 15	3	104	8	5.5
Great Fingall..	1150	....	....	12 × 12 10 × 10	3	106	8	7.0

TABLE XLI.—APPROXIMATE LIFE OF STAMP-MILL PARTS[51]

Part	Material	Dimensions	Weight, lb.	Life
Camshaft.....	Wrought iron	5.36″ diam.	1050–1085	4 yrs.
Cam.....	Chrome steel	2.5″ face × 35″ hub	262	no data
	Cast iron	12″ diam. × 5.75 in.	256	2 yrs.
Tappet.....	Chrome steel	9″ × 12″	150	no data
	Cast iron	—	132	20 mths. & up.
Stem.....	Wrought iron	3.125″ × 15′	380–390	4 mon. bet. breakages..
Bosshead.....	Cast iron	9″ × 18″	236	4–12 yrs. average
Shoe.....	Chilled cast iron	9″ × 8″	145	60–90 days
Die.....	Hard cast iron	9″ × 5.5″	110	30–35 days
Mortar.....	Cast iron....	—	5500	3 years
Screen.....	Cold-rolled re-annealed O. H. steel..	9 × 50 × 0.035	4	10–16 days
Plate.....	Lake Superior copper.....	4′ 5″ × 12′ × 0.125″	320	

NOTE.—The thickness of mortar-box ends has been since increased and a longer life is anticipated.

of South African practice in this respect are given in Table XXXVII, of American practice in Tables XXXVIII and XXXIX and of Western Australian practice in Table XL.

Data Giving the Approximate Life of Stamp-Mill Parts and referring to Homestake practice are given in Table XLI.

Comparative Costs for **Breaking, Crushing** and **Grinding** at various plants in the United States and Canada are given in Table XLII.

TABLE XLII.—COST OF ORE MILLING[52]

Plant	Breaking, cents per ton	Crushing, cents per ton	Grinding, cents per ton	Method
Homestake	6	34.6	25.7‡	Stamps, pans and tube mills
Goldfield Con.	3.8	13.4	10.6 Chilean 16.6 tube	Stamps, Chilean and tube mills
West End Con., Tonopah	10.3*	24.8	49.2	Stamps and tube mills
Hollinger	7.3	18.1	27.7	Stamps and tube mills
Wonder, Nev.	12.1*	29.5	41.8	Stamps, tubes and high-speed Chileans.
Liberty Bell	7.25	19.08	7.69	Stamps and tube mills
Independence	16.33	52.17†	....	Rolls and high-speed Chileans
Nevada Hills	7.1	24.7	29.3	Stamps and tube mills
Black Oak	5.74	11.26	7.99	Stamps and tube mills
Sta. Elena	....	21.22§	....	Slow-speed Chileans
La Union	....	21.46§	....	Slow-speed Chileans.

* Includes conveying. † Separate treatment plant. Cost includes rolls and Chileans. ‡ Only part of material reground. Cost per ton reground. § Includes both crushing and grinding.

Power Used for Stamp Mills may be calculated from the diagram given in Fig. 89. For example: To find horsepower to drive a 10-stamp mill, each of 900 lb., at 90 drops per minute, follow up the vertical line commencing at 900 until it intersects one of the diagonal lines representing the height of drop. The horsepower required for a 7-in. drop will be found to be 17.5, and for an

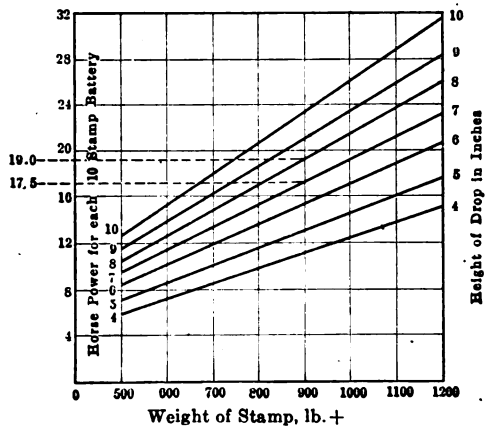


FIG. 89.—Power required for stamps, at 90 drops a minute.[53]

8-in. drop 19.0. These figures do not include total friction and should be multiplied by 1.2 to give an approximation of the total power required.

The Tremain Stamp (Fig. 90) is a self-contained unit suitable for small properties. Two stamps are used in each battery, the ends of which terminate in pistons. Steam is admitted under the piston to raise the stamp and also assists in the downward movement, thus increasing the effective stamp weight. The stamps are of comparatively light design (about 500 lb. each) and can be arranged to drop up to 200 times a minute. Discharge is through front and side screens.

Heavy-Duty Steam Stamps (Fig. 91) are of limited application and have only been adopted in exceptional cases. The two types

in general use on copper properties are designed on either the simple- or compound-engine system. A heavy feed is necessary, and actual duty is consequently low. A light feed predisposes toward piston-rod breakage. Del Mar quotes the usual stroke

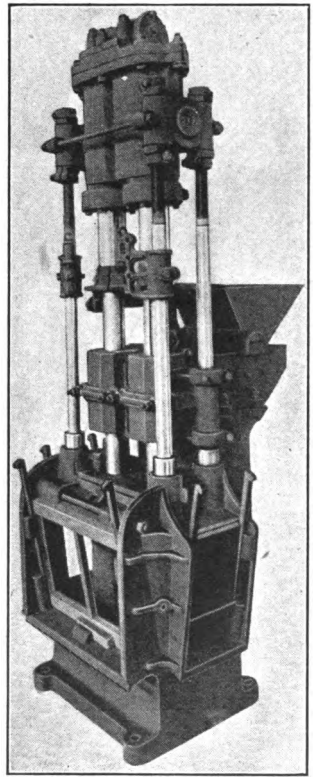


FIG. 90.—Tremain steam stamp.[11]

in the Lake Superior district, where these stamps are mostly used, as 24 to 25 in. on 4-in. material, giving a capacity of from 2 to 3 tons per horsepower day through a $\frac{5}{8}$-in. mesh screen. Shoes lasting from 4 to 7 days weigh from 780 to 800 lb. when new, and 380 to 400 lb. when rejected. Dies last 7 months to a year, and

weigh 800 lb. when new and 500 lb. when rejected. Height of discharge would seem to vary between 9 and 16 in., but the former

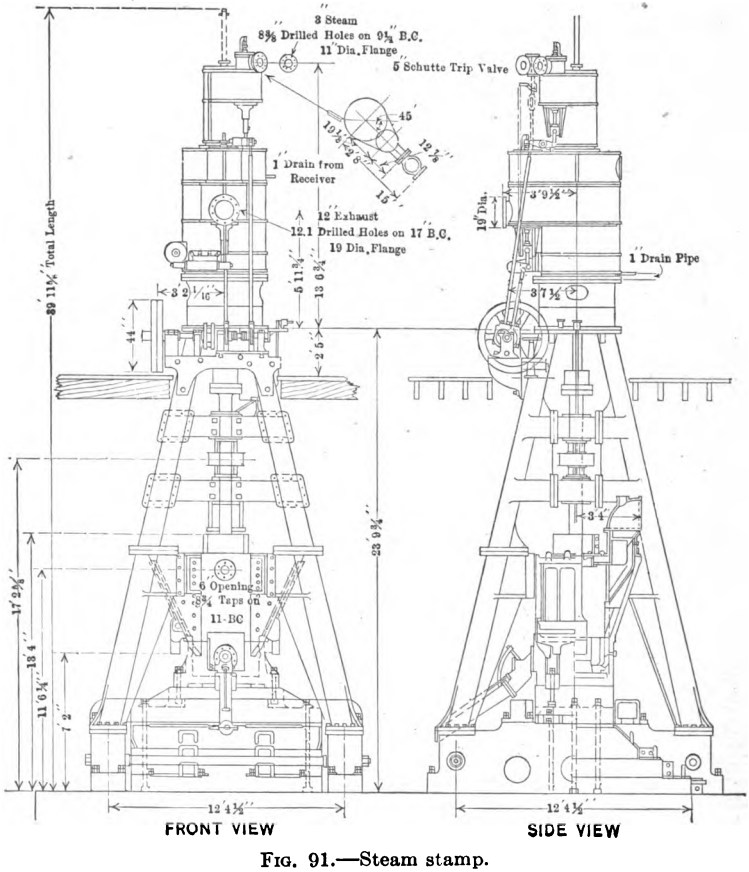


FIG. 91.—Steam stamp.

figure is probably nearer the average. The stamp head weighs 6650 lb. and the stamp is dropped with a frequency of 108 per minute.

SECTION IX

BALL MILLING

The **Krupp Ball Mill** (Fig. 92) consists of a narrow cylinder lined with steel plates arranged in steps. The grinding and

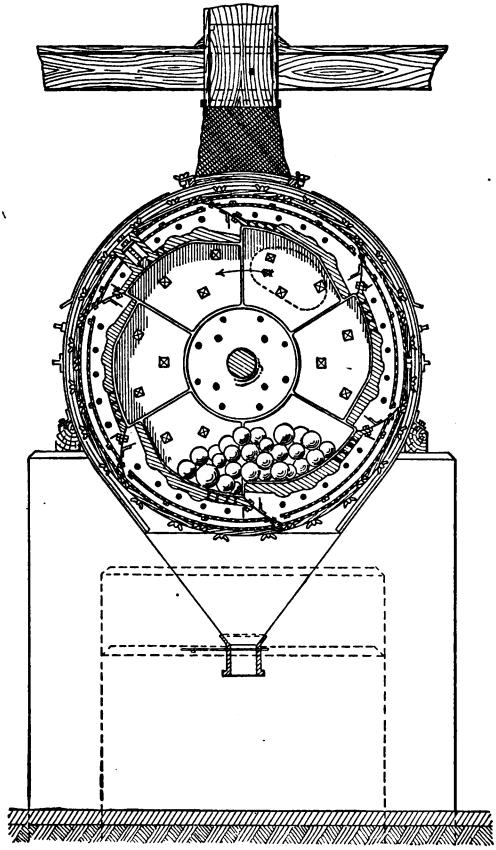


FIG. 92.—Krupp dry-crushing ball mill.

crushing are done by steel balls and the ground material passes

through tapered perforations, usually ⅜ in. to 1 in. diameter, in the grinding plates. A coarse punched screen, with holes about ¾ in. in diameter, and placed outside the lining plates,

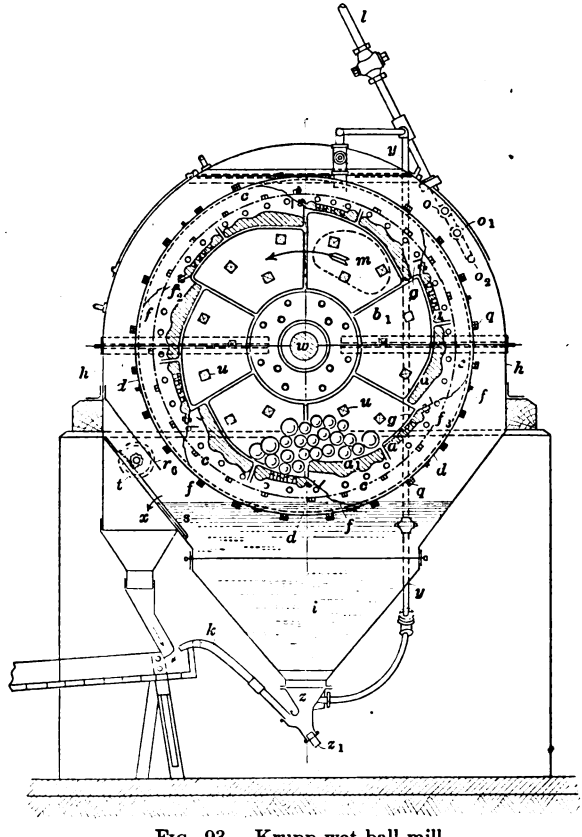


FIG. 93.—Krupp wet ball mill.

a, Perforated grinding plates; a_1, Flange bolts; *b*, Side liners on the feed side; b_1, Side liners, rear side; *u*, Side-plate bolts; *c*, Inner coarse screen; c_1, Inner screen bolts; *d*, Fine screen (frame with wire cloth); *q*, Bolts for screen frame; *f*, Return scoops; f_2, Return screen; f_3, Bracket for coarse screen; *h*, Sheet-iron casing; *i*, Spitzkasten; *k*, Discharge nozzle; *l*, Supply pipe for fresh water; *o*, o_1, o_2, Roses; *p*, Stopcocks; *s*, Slide; r_6, Slide-racking gear; *t*, Hand wheel for slide-racking gear; *m*, Manhole; *w*, Main shaft; *x*, Overflow for slime; *y*, Supply pipe for ascending water; *z*, Discharge box; z_1 Plug.

makes a further classification; and the fines pass to the wire cloth arranged in frames and covering the periphery of the mill. The oversize passes back into the mill through a perforated plate

TABLE XLIII.—KRUPP BALL MILL DATA[7]

Mine	No. of mill	Rev. per min.	Horse power	Screening	Weight of balls	Tons per day
Associated Northern...	5	25	16.6	700 × 28 (26 × 27) (I. S. W. G.)	2260–2400	40
Kalgurli.....	5	25	.....	32 × 32 × 30 (I. S. W. G.)	....	40
Associated.....	5	25	16.8	30 × 30 × 30 (I. S. W. G.)	2240	...
South Kalgurli.....	5	25	.....	.....	2352	40
	8	24	57	700 × 26 (26 × 27)(I. S. W. G.)	3472	110
Gt. Boulder Propy.....	8	23	60	30 × 30 × 28 (I. S. W. G.)	4256	90

TABLE XLIV.—KRUPP BALL MILL PRODUCTS[7]

Mine	Associated Northern	Kalgurli	South Kalgurli	Great Boulder Propy.
Screening	700 × 28 (I. S. W. G.), Per cent.	32 × 32 × 30 (I. S. W. G.), Per cent.	Per cent.	30 × 30 × 28 (I. S. W. G.), Per cent.
+ 30	....	.....	.....	0.22
+ 40	11.6	7.96	8.80	5.21
+ 60	13.6	10.92	12.79	15.74
+ 80	10.2	.....	7.01	6.64
+ 100	7.3	21.67	7.20	8.07
+ 120	2.0	6.65	3.45	1.67
+ 150	1.0	5.00	2.45	6.32
− 150	54.3	47.80	58.30	56.13
	100.00	100.00	100.00	100.00

placed beneath the steps. Data dealing with dry-crushing ball mills may be found in Table XLIII and details of typical grading analyses in Table XLIV.

Dry Crushing with **Krupp Ball Mills** has been largely practised in Western Australia on sulpho-telluride ores. The reason for the adoption of dry crushing lies in the fact that roasting follows crushing and precedes cyanide treatment.

The **Wet-Crushing Krupp Ball Mill** (Fig. 93) operates in the same manner as the dry crusher. In addition, a water spray

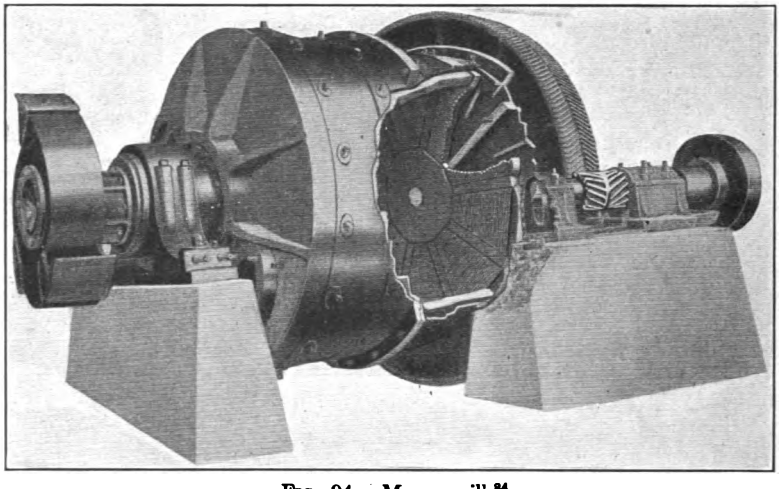


FIG. 94.—Marcy mill.[84]

delivers over the whole width of the fine screening just after the latter, in revolution, leaves the water surface in a trough placed underneath the mill, and forming the lower part of the casing. This trough consists of a spitzkasten with gooseneck discharge and adjustable overflow level. Water is caught up during the revolution of the mill. Crushing, screening over an exceptionally large area, and return of oversize are all performed simultaneously.

The **Marcy Mill** (Fig. 94) is of the short cylindrical type with comparatively large diameter. Feed is delivered through the usual trunnion-type feeder and delivery takes place through a

grizzly-bar screen, covering the entire discharge end of the mill. The pulp delivered into the space behind the grizzly bars is lifted with spirals and discharged through the trunnion. The mill is designed to take 3-in. or 4-in. material and crush this through ¼ in. or ⅛ in. in one operation. With a closed-circuit return system and outside classification the mill can be arranged to crush as fine as 100 mesh. Steel-ball consumption varies from 0.5 to 1.7 lb. per ton crushed.

The usual type **Ball Mill,** used widely for reduction from crushed rock size to 80 mesh or so, consists of a short cylinder

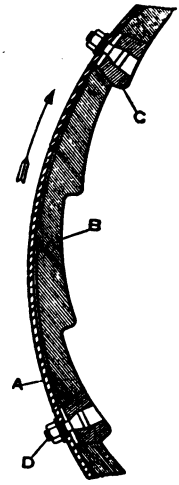


FIG. 95.—Wave-type ball-mill lining.[31]

driven by spur gearing. The machine is operated at maximum efficiency in closed circuit with a classifier. The wave type of lining suitable for this mill is shown in Fig. 95 and is constructed of plates with beveled ends to match the key liners *C*, the latter being alone bolted to the mill shell by means of countersunk head bolts *D*.

Standard Sizes of Ball Mills are given in Table XLV, with approximate capacities in Table XLVI. Methods of drive are shown in Figs. 96*a*, 96*b*, and 96*c*.

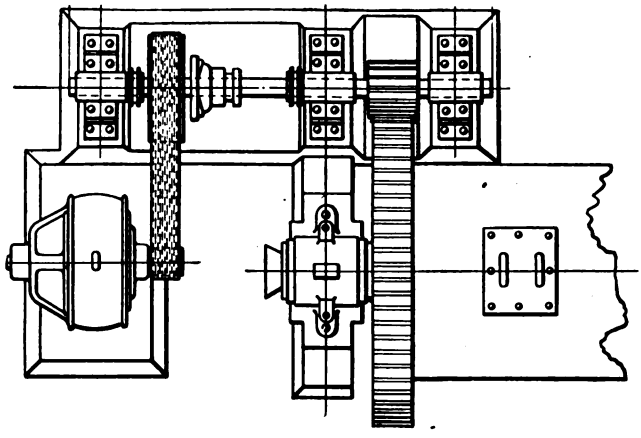


FIG. 96a.[31]

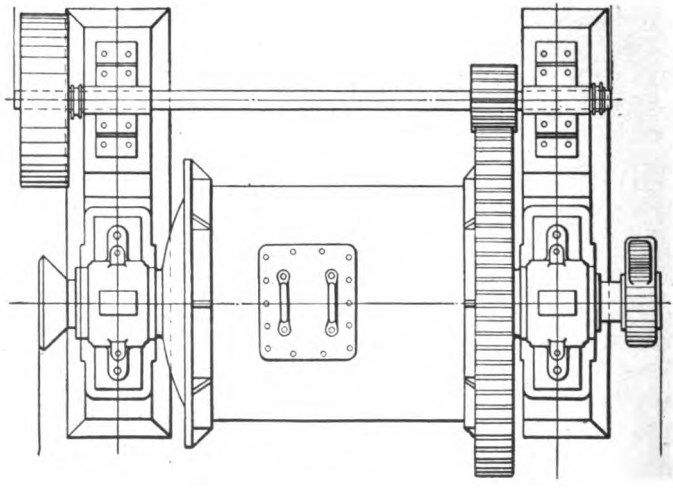


FIG. 96b.[31]

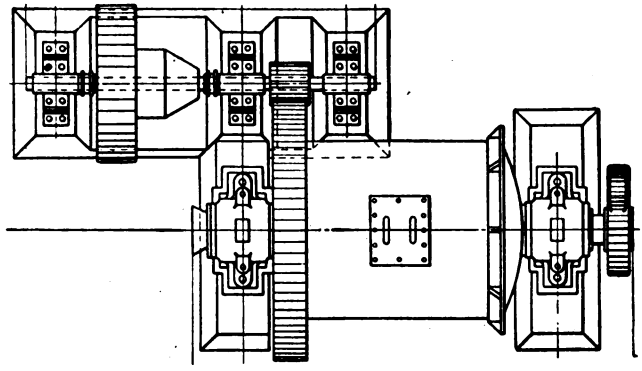


FIG. 96c.—Methods of driving ball mills.[31]

TABLE XLV.—STANDARD BALL MILLS[31]

Size of mill in feet	Size of pulley in inches	Rev. per min.		*Horse power	†Ball charge in tons	‡Weight of mill in pounds
		Pulley	Mill			
3 × 3	36 × 6	168	32	6	0.5	5,000
3 × 4	36 × 6	168	32	8	0.7	5,500
3 × 5	36 × 8	168	32	10	0.9	6,000
4 × 3	36 × 8	171	29	14	1.2	9,500
4 × 4	36 × 10	171	29	18	1.5	10,000
4 × 5	36 × 10	171	29	22	1.8	10,500
4 × 6	42 × 10	171	29	26	2.1	11,000
5 × 3	54 × 10	171	26	24	2.0	19,000
5 × 4	54 × 10	171	26	30	2.6	19,500
5 × 5	54 × 12	171	26	36	3.2	20,000
5 × 6	54 × 14	171	26	42	3.8	21,000
6 × 3	66 × 12	171	23	50	3.0	35,000
6 × 4	66 × 14	171	23	62	4.0	36,000
6 × 5	66 × 16	171	23	74	5.0	37,000
6 × 6	66 × 18	171	23	86	6.0	38,000

*The figures for power are for the mill while running, with balls carried up to center of mill. The power will vary with the ball charge, and the power to start the mill will be approximately double that consumed while the mill is running.

†The ball charges are approximate only, as they will vary with the thickness of the lining and the exact point at which it is found desirable to carry the balls under the particular conditions.

‡The weights given are for the mills complete with head liners and driving shaft with pinion, bearings and plain pulley; but do not include shell liners.

TABLE XLVI.—AVERAGE BALL-MILL CAPACITIES[31] IN TONS PER 24 HR.

Size of mill, ft.	2-in. feed				1-in. feed				½-in. feed		
	Mesh product										
	10	20	40	60	10	20	40	60	20	40	60
3 × 3	...	...	...	...	15	12	8	5	13	9	6
3 × 4	...	...	...	...	19	15	10	6½	16	11	8
3 × 5	...	...	...	...	23	18	12	8	19	13	10
3 × 6	...	...	...	...	26	20	14	9½	21	15	12
4 × 3	60	40	30	20	65	44	33	22	50	36	24
4 × 4	72	50	37	25	78	55	40	27	60	43	30
4 × 5	85	60	44	30	90	66	48	33	72	52	36
4 × 6	95	70	50	35	105	77	55	38	85	60	42
4 × 8	105	80	57	40	115	88	63	44	95	68	48
5 × 3	90	60	45	30	100	66	50	33	72	54	36
5 × 4	110	90	60	40	120	100	66	44	108	72	48
5 × 5	125	110	75	50	135	121	83	55	130	90	60
5 × 6	135	130	90	60	150	143	100	66	155	108	72
6 × 3	160	100	75	50	175	110	82	55	120	90	60
6 × 4	210	135	100	75	230	150	110	82	160	120	90
6 × 5	250	165	125	100	275	180	137	110	200	150	120
6 × 6	280	200	150	125	310	220	165	138	240	180	150

The **Hardinge Mill** (Fig. 97) consists of three sections. After the usual type of trunnion bearing the mill consists of a short conical section showing a sharp drop from feed entrance to maximum mill radius. A cylindrical section of varying length then follows and toward the discharge end connects with a comparatively long and conical section sloping, at considerably less pitch than at the feed end, toward the discharge. In principle it has been claimed that a separation and grading of the balls in sizes, resulting in a corresponding graduation in the force of the crushing blow, occurred in the mill to the extent that single-passage crushing might be advocated. The successive reduction of the ore particle from rock to sand or gravel to slime was pictured as an actual achievement and as a logical departure from the normal current practice with cylindrical mills. These views have

been proved somewhat fallacious. The balls do not segregate to the extent assumed, and careful investigation has proved that the discharge sections may consist of balls of a larger average size than the medium of the balls used in the mill. A. F. Taggart (Trans. A. I. M. E., Sept., 1917) has also demonstrated that efficient Hardinge-mill operation cannot be effected by single-passage

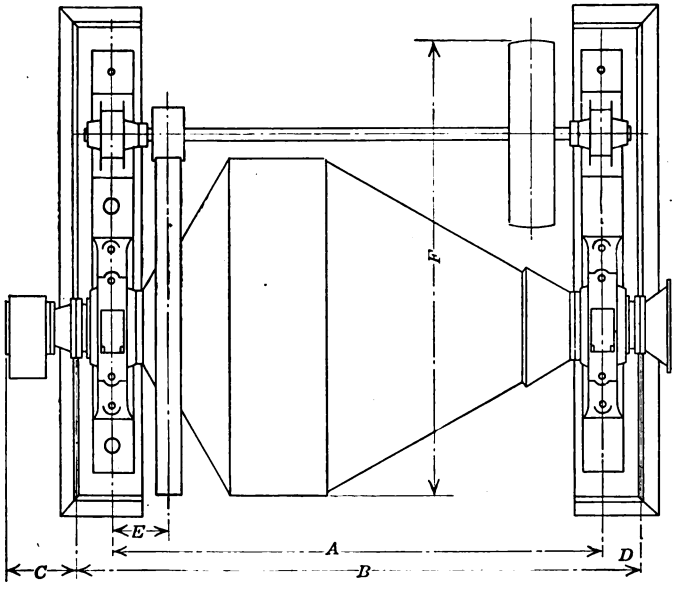


FIG. 97.—Hardinge pebble or ball mill.[85]

grinding, and a return classification system is indicated as desirable, as with the cylindrical mill. This would seem effectively to dispose of the "graduated blow" theory.

Standard Sizes of Hardinge Mills are given in Table XLVII. A summarized statement of data, consisting of the average performance of a number of these machines, is given in Table XLVIII.

TABLE XLVII.—STANDARD SIZES OF HARDINGE MILLS[85]

Mill, ft. in.	A, ft. in.	B, ft. in.	C, in.	D, in.	E, in.	F, ft. in.	Size of pulley, in.	Gear ratio
4½×13	6– 6	7–5	11	5	9¼	6–1½	30×10	82:17
6×16	8– 5	9–9⅜	13¼	6½	12½	8–1⅜	40×12	88:17
6×16	8– 5	9–10	16¼	6½	13¾	8–2⅜	42×14	88:17
6×22	8–11	10–3⅜	13¼	6½	12½	8–1⅜	40×10	88:17
6×48	11– 1	12–5⅜	13¼	6½	12½	8–2⅜	42×12	88:17
6×72	13– 1	14–5⅜	13¼	6½	12½	10–2¼	42×14	118:17
8×22	11– 2½	12–6⅞	13¼	6½	12½	10–2¼	42×12	118:17
8×22	11– 3¾	12–8¾	16¼	6½	13¾	10–2¼	42×14	118:17
8×30	11–10½	13–2⅞	13¼	6½	12½	10–2¼	42×14	118:17
8×30	11–11¾	13–4¾	16¼	6½	13¾	10–2¼	42×14	118:17

NOTE: Mills marked with a subscript "s" have 15-in. dia. bearings; other 6-ft. and 8-ft. mills have 13-in. dia. bearings.

Pulley size given for 4½ ft. × 13-in. mill is for Ball Mill only; 28-in. × 8-in. pulley furnished with Pebble Mill.

For position of over-all dimensions, A to F, see Fig. 97.

TABLE XLVIII.—HARDINGE-MILL DATA[54]

	6-ft. × 16-in. ball mill	8-ft. × 22-in. pebble mill
Average maximum size feed, mm	44.5	9.7
Average size feed, mm	9.0	1.26
Average maximum size discharge, mm	6.0	1.5
Average size product, mm	0.37	0.14
Average per cent.—200-mesh in discharge	28.9	37.0
Average per cent.—200 mesh in discharge—no slope	.....	44.3
Average per cent.—200-mesh in discharge 0.5"–4" slope	.....	31.6
Reduction ratio range	7–67	6–15
Reduction ratio, average	39.6	0.8
Average size of discharge, mm.—no slope	.....	0.10
Average size of discharge slope 0.5" to 4"	.....	0.17
Average tonnage	203	110
Average tonnage at no slope	.....	85
Average tonnage at 0.5" to 4" slope	.....	128
Average H.P.	35.06	35.6
Average ball or pebble charge, tons	4.0	4.5
Average ball or pebble cons., lb. per ton	0.51	1.94
Average per cent. water in feed	60	58.7
Average revolutions per minute	28	27.8

Variable Pulp-Level Discharge in ball mills is provided for in the Allis-Chalmers Ball Granulator. In this machine the level of the pulp may be raised to a point approximately halfway between the trunnion and the periphery, or lowered to the periphery. This is accomplished by closing outer rings of openings behind the screens and inside the mill end (see Fig. 98). Radial ribs are provided to elevate the crushed pulp and dis-

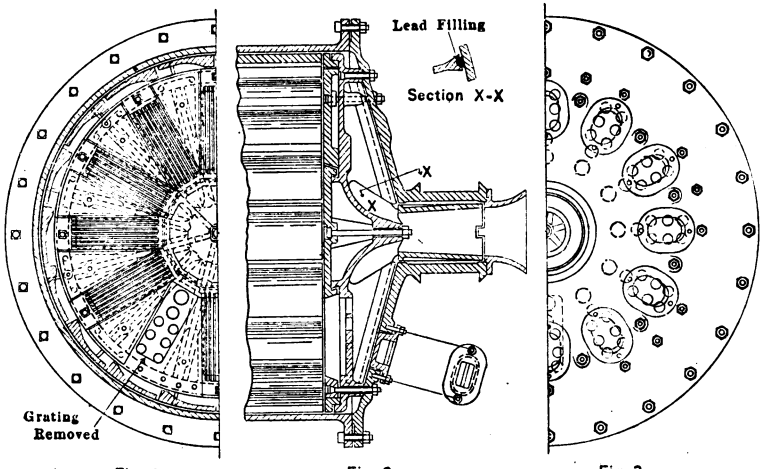


Fig. 1 Interior View Showing Screen

Fig. 2 Longitudinal Section Showing Screen and Discharge End Head

Fig. 3 Exterior View Showing Discharge End Head, and Hand Holes

FIG. 98.—Ball-granulator variable-discharge screen.[11]

charge it through the trunnion. Sundry data dealing with this type of mill are given in Table XLIX.

Screen Analysis of Ball-Granulator Products and other information is given in Fig. 99 and refers to the operation of this type of machine at the property of the Engles Copper Mining Co., Keddie, Cal., on a hard diorite ore.

The **Abbé Spiral Discharge and Screen** for ball mills (Fig. 100) consists of a convolute with diameter decreasing toward the center line of the mill. The spiral is lined with screening in the manner shown. The material is discharged from the mill at the outer

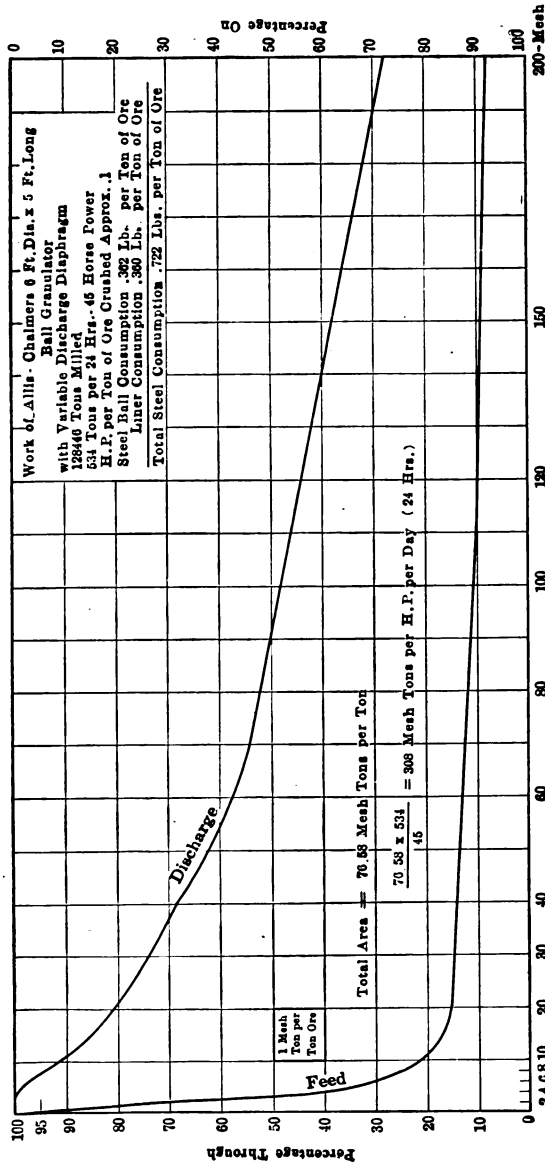

Work of Allis-Chalmers 6 Ft. Dia. x 5 Ft. Long
Ball Granulator
with Variable Discharge Diaphragm
128440 Tons Milled
534 Tons per 24 Hrs.-45 Horse Power
H.P. per Ton of Ore Crushed Approx. 1
Steel Ball Consumption .362 Lb. per Ton of Ore
Liner Consumption .360 Lb. per Ton of Ore
Total Steel Consumption .722 Lbs. per Ton of Ore
Discharge
Feed
1 Mesh Ton per Ton Ore
Total Area = 70.53 Mesh Tons per Ton
70.53 x 534 / 45 = 308 Mesh Tons per H.P. per Day (24 Hrs.)
Percentage Through
Percentage On
200-Mesh

FIG. 99.—Example of ball-granulator screen analysis.[11]

TABLE XLIX.—STANDARD SIZES OF BALL GRANULATORS[11]

Size of Mill		Weight of charge of balls, lb.	Capacity in tons, 3 in. to 50 mesh	Capacity in tons, 1½ in. to 20 mesh	H.p. required to run	Size of motor recommended	Revolutions per minute of mill	Speed of countershaft		Size of pulley	Approximate weights complete	
Dia.	Length							Spur gear drive	Wuest gear drive		With pulley and spur gear drive*	With Wuest gear drive†
4′–0″	4′–0″	5,500	45	55	25	40 H.p.	32.5	170	580	30″×15″	16,500	18,500
5′–0″	4′–0″	9,000	65	90	40	60 H.p.	28.0	145	430	54″×17″	29,000	29,000
5′–0″	5′–0″	11,000	85	110	50	75 H.p.	28.0	145	430	54″×17″	32,000	31,000
6′–0″	4′–0″	13,500	100	135	60	85 H.p.	24.0	145	435	72″×17″	37,500	35,000
6′–0″	5′–0″	17,000	125	165	80	100 H.p.	24.0	145	435	72″×17″	40,000	37,000
6′–0″	6′–0″	20,000	150	200	100	125 H.p.	24.0	145	435	72″×17″	42,000	39,500
7′–0″	5′–0″	23,000	170	225	110	125 H.p.	20.0	115	390	84″×19″	50,500	47,500
7′–0″	6′–0″	27,000	200	270	135	150 H.p.	20.0	115	390	84″×19″	53,000	49,500
8′–0″	5′–0″	30,000	225	300	150	175 H.p.	18.0	108	295	84″×21″	62,500	61,000
8′–0″	6′–0″	36,000	270	360	180	200 H.p.	18.0	108	295	84″×21″	65,500	63,500

* Includes liners, shell, heads, bearings, countershaft and pulley, etc., but without balls

† Includes liners, shell, head, bearings, countershaft, etc., but without motor or balls.

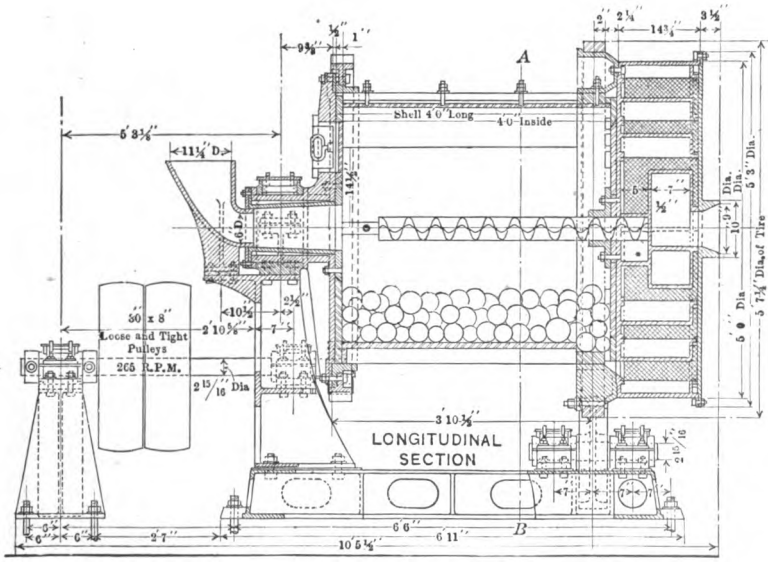


FIG. 100.—Abbé ball mill.[55]

point in the spiral and is sifted by the screening. The oversize is automatically returned by the conveyor as shown in Fig. 101. The screened undersize follows the course of a second spiral underneath the screening, and is discharged from the center.

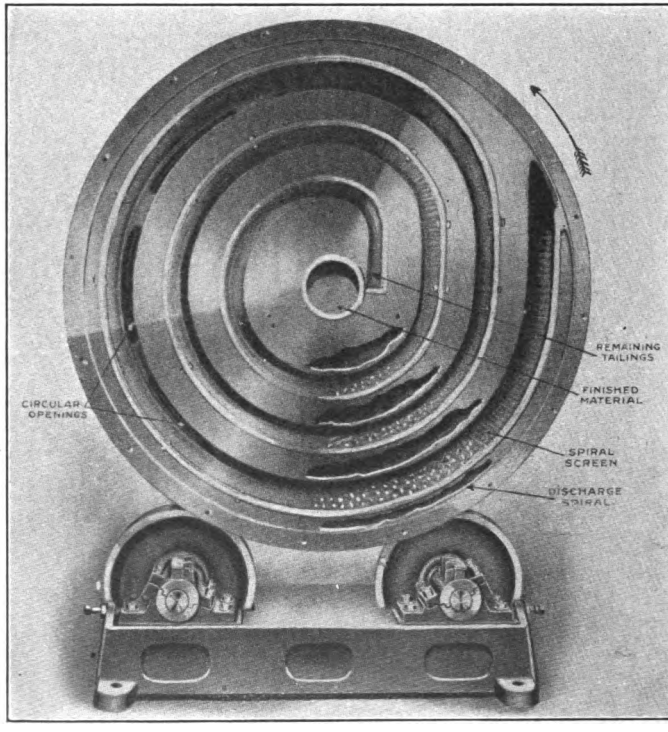


FIG. 101.—Abbé ball-mill discharge screen.[55]

In the **Eccleston Peripheral Discharge** for ball mills (Fig. 102) screening is accomplished by means of grizzly-type liners which are adjustable, as regards spacing, throughout the length of the mill. In the type illustrated the interior of the mill is smooth, so that grinding action would result mainly from the abrasion of the sliding balls in contact with the ore.

Preliminary Load of Balls in a Ball Mill may be arranged to correspond with normal sizes found under ordinary operating conditions, but tests have shown that this is not always necessary and that the balls soon wear, so that a mixture of all sizes is produced. In starting up a 5-ft. Krupp mill it is customary to make up the load as follows:

45 per cent. by weight of large balls weighing about 18 lb.
20 per cent. by weight of medium balls weighing about 9 lb.
35 per cent. by weight of small balls weighing about 5 lb.

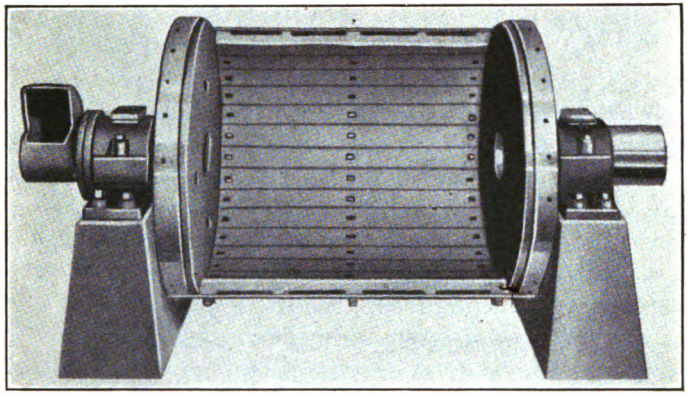


FIG. 102.—Peripheral-discharge ball mill.[56]

Sizes and weights of forged-steel balls in such mills are given in Table L. A special small cast-steel ball of ⅞ in., 1½ in., or 2 in. diameter is made by the Allis-Chalmers Co., and weighs 5 per cent. less than a hammered-steel ball of corresponding size.

Size of Feed in Ball Milling should be in direct ratio to the diameter of the mill and the size of the balls used for grinding. In some instances a feed up to 6 in. in diameter has been effective where it has been found that fracture of the smaller rock was accomplished by the larger portions of the feed, but this is generally only feasible with a friable ore and a large mill. A uniform feed of 2-in. material, or thereabouts, usually gives best results with an average-size (5-ft. diam.) mill.

TABLE L.—SIZES AND WEIGHTS OF FORGED-STEEL BALLS[31]

Diameter of balls, in.	Weight of one ball, lb.	Approx. number of balls in 2000 lb.	Diameter of balls, in.	Weight of one ball, lb.	Approx. number of balls in 2000 lb.
⅞	0.10	20,250	3	3.96	505
1	0.15	13,480	3½	6.34	315
1¼	0.29	6,230	4	9.48	211
1½	0.50	4,010	4½	13.50	148
1¾	0.79	2,520	5	18.54	108
2	1.18	1,690	5½	24.65	81
2½	2.31	860	6	32.00	63

Low Feed Tonnage, in an attempt to make the mill do the work in a single passage, results in the grinding of a considerable proportion of the material beyond the required limit. In the case where all-sliming is being practised it results in a decreased duty, on account of the formation of slime in the first half of the mill as a result of an almost immediate reduction of the finer material present in the feed. A further disadvantage and loss from insufficient feed is seen in high ball consumption and excessive wear of the liners at the discharge end of the mill. This objection does not apply to the conical mill, where little effective grinding is performed at the discharge end and where wear of liners is proportionally light. **Ball-Mill Feed** should be increased to a point showing maximum grinding with minimum ball and liner consumption. Ample feed insures high duty, because, under such conditions, there is always a layer of sand between the balls, even at the discharge end. It prevents excessive ball and liner wear by cushioning the blow between ball and ball, and ball and liner, with a layer of material to be crushed. The disadvantage of high feed is seen in the necessity for a correspondingly high-duty classifying and return system; but, as elevating classifiers perform this operation at an insignificant cost, the final advantage is always in favor of an ample ball- or tube-mill feed, with a closed-circuit return system.

The **Close-Circuit System** of tube and ball milling has been shown by numerous investigations to give maximum efficiency, largely because it would be impossible to adjust any mill of this type to exactly fulfill momentary duty. Single-passage crushing to exact requirements is not practicable, because the apparatus has no flexibility and would be unable to cope with slight variations in tonnage with any prospect of efficient operation. The reason for improved output with closed-circuit grinding is due to the fact that provision is made for the removal of material already ground sufficiently fine, and at the earliest possible moment. It permits an effective visible control of operations in the quantity of classifier return, with the result that, at the outset of an accumulation in the circuit, the necessary steps can immediately be taken (by increasing pebble load or altering moisture conditions in the mill feed) to increase the grinding without stopping the feed or permitting any material alteration in the composition of the final product from the classifiers. The grates in certain types of ball mills were originally constructed with the idea of screening the pulp; but it is now becoming realized that if any ore particles have reached the discharge end of the mill without being ground to the required size nothing will be gained by retaining them there, where no provision exists to crush or grind more than the normal flow. Grates, when used, are now provided with larger openings, which permit the passage of partly-ground ore; and more reliance is placed on the efficient operation of a mechanical classifier to separate the finished from the unfinished product, and to return the latter to the feed end of the mill. If the grate were used as a screen, and if any particles of ore in the pulp were insufficiently ground when the screen were reached, there would be an immediate accumulation, which would interfere with the operation of the mill. The work of grinding increases as reduction in particle size proceeds, so that the smaller duty in the primary stages of grinding in a cylindrical mill is counterbalanced by the provision of an ample pulp return to the feed end.

Comparisons Between the Work of Crushing in Wet or Dry Ball Mills show increased duty with the use of water. At the

same time wet crushing involves shorter life for liners and increased consumption of steel balls. In summing up it may be said that the advantages are in favor of wet crushing, where possible, because first cost is practically the same for either type of mill, so that depreciation and capital charges on wet crushers are less per ton crushed than in the case of the dry type. Power is also practically the same whether the mill is crushing wet or dry, and this fact also influences the comparison in favor of wet crushing.

SECTION X

GRINDING IN CHILEAN, HUNTINGTON, AND SIMILAR TYPES OF MILLS

The **Chilean Mill** (Fig. 103) consists of a shallow circular trough of a diameter which denotes the size of the mill. Large-

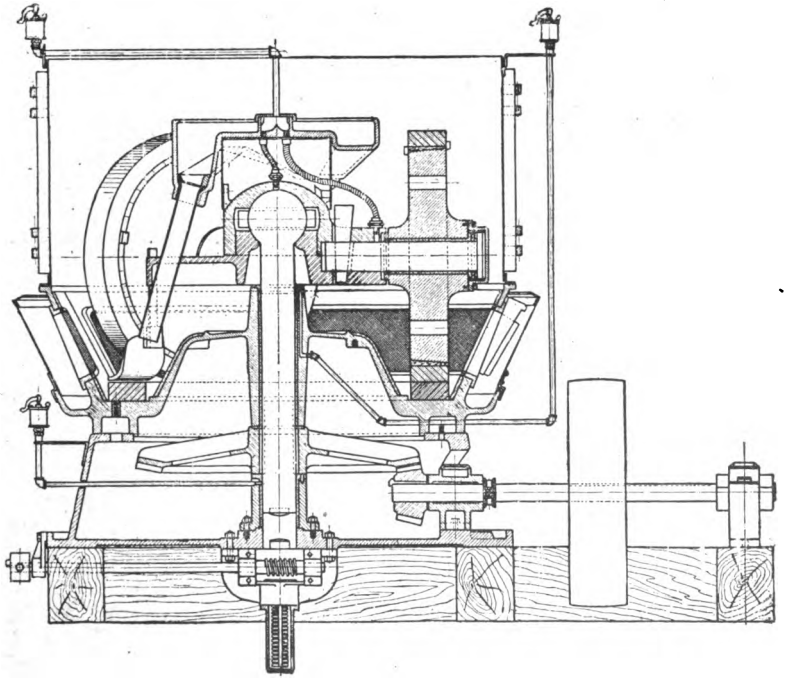


FIG. 103.—Chilean mill.[18]

diameter rollers of comparatively narrow face are revolved in the trough, and the ground product is discharged through

appropriate screening. Details of construction will be found in Table LI, and operation in Table LII.

The **Lane Mill** is an improved Chilean mill operating at a slow speed with an overflow discharge generally without screening. A bed of ore is kept between tires and die, and the mill is surmounted with a weight tank, which, when filled with rock or other heavy material, adds to the crushing power of the tires. Comminution is insured by a combination crushing, grinding, and abrasive action. Construction details are given in Table LIII, general operating data in Table LIV, and screen analyses in Table LV.

TABLE LI.—CHILEAN-MILL DATA[18]

Size	Weight of heaviest piece	Horsepower	Speed of mill r.p.m.	Total weight
5′	8,000 lb.	30	35	38,000 lb.
6′	11,000 lb.	40	30	50,000 lb.
8′	45,000 lb.	60	20	125,000 lb.

TABLE LII.—CHILEAN-MILL PERFORMANCE[57]

Ore	Quartz-porphyry
Screen	1½ mm. round hole punched
Die weight	1722 each when new, total 2526 lb.
Tons feed per 24 hours	237
Per cent. solids in feed	24.9
Ratio solids to water	1 to 3
Tons dry feed per H.p. day	6.485
H.p. per hr	36.70
Wear per ton crushed:	
dies	0.11533 lb.
tires	0.29153 lb.

TABLE LII.—(*Continued*)

Screening analysis				
Mesh	Opening, in.	Opening, mm.	Percentage by weight	
			Feed, Per cent.	Discharge, Per cent.
+4	0.185	4.699	0.94	.....
+6	0.131	3.327	7.71	.....
+8	0.093	2.362	16.49	0.05
+10	0.065	1.651	11.84	0.10
+14	0.046	1.168	8.98	1.07
+20	0.0328	0.833	7.18	3.63
+28	0.0232	0.589	7.43	7.75
+35	0.0164	0.417	6.04	9.08
+48	0.0116	0.295	4.80	9.38
+65	0.0082	0.208	3.89	8.53
+100	0.0058	0.147	3.76	8.95
+150	0.0041	0.104	2.58	6.23
+200	0.0029	0.074	1.82	4.21
−200	.....	.....	16.54	41.02
			100.00	100.00

TABLE LIII.—LANE-MILL CONSTRUCTION DETAILS[58]

Size of mill	7 ft.	10 ft.
Capacity, tons per 24 hours	20–30	40–50
Speed of mill, r.p.m.	11	8
Speed of pulleys, r.p.m.	290	260
Size of pulleys	24″ × 10″	24″ × 12″
Power required to operate	12 h.p.	20 h.p.
Net weight	17,000 lb.	26,450 lb.
Weight of heaviest piece (i)	1,240 lb.	1,240 lb.
Height	6 ft.	6 ft.
Floor space occupied (ii)	9 ft. × 15 ft. 2 in.	12 ft. × 18 ft.
Distance between feed and discharge levels (iii)	1 ft. 10 in.	2 ft. 2 in.
(iv)	6 ft. 5 in.	6 ft. 9 in.
Proper size of feed	1 in.	1 in.

(i) For mule-back transport, 250 lb.
(ii) With overhead drive reduce larger dimenstion by 6 ft.
(iii) When discharging into pan
(iv) When discharging into overhead distributor

TABLE LIV.—LANE SLOW-SPEED CHILEAN-MILL DATA[59]

Data	1	2	3	4	5	6	7
Size of mill, in feet.....	7	10	10	7	7	10	10
Time in actual operation	A	2 yr.	3 yr.	G	8 mo.	18 mo.	K
Capacity, tons in 24 hours.....	24	40	55	14	20	30	45
Horsepower required....	7	10	12	?	7	10	8
Gallons of water used per minute.....	M	B	D	?	7	15	few
Height of discharge, ins.	4	4	7	?	6	7	7½
Mesh of screen, if any...	30	8	..	...	..	..	..
Character of ore.....	med.	hard	soft	hard	?	hard	med.
Mesh of discharged pulp.	44	40	E	H	I	J	L
Time shut down for repairs.....	5 dy. p. mo.	C	2 hr.	?	..	..	..
Cost of repairs, per ton..	5c.	5c.	F	?	3c.	5c.	4c.
Size of feed, inches.....	½	1	½	½	½	1	1

REMARKS

A—Total crushed in 8500 tons.
B—Same as a 10-stamp mill.
C—Repairs made while cleaning up.
D—Have an unlimited supply of water. No record kept of consumption.
E—90 per cent. is minus mesh.
F—Lower than stamps.
G—90 shifts to date.
H—80 per cent. is minus 100 mesh.
I—95 per cent. is minus 100 mesh.
J—68 per cent. is minus 100 mesh.
K—Milled a total of 7000 tons to date of writing.
L—73 per cent. is minus 100 mesh.
M—16,000 gal., per 24 hours.
Where not otherwise stated, no screens were used.

1. In Arizona. Considers Lane better than stamps. Feed direct from crusher to Lane mill.
2. In southern Nevada. An excellent fine grinder and amalgamator.
3. In Idaho. An ideal amalgamator; 81 per cent. caught in the mill itself.
4. In California. Not satisfied. Grinds too fine. Capacity limited.
5. In Idaho. Better than stamps as a crusher; less wear and less water needed.
6. In Nevada. Ore extremely hard; free milling; 80 per cent. of pulp through 60 mesh.
7. In Texas. An excellent machine for fine grinding.

The above data were obtained in August, 1912, from correspondence with companies actually operating the mill.

TABLE LV.—LANE-MILL SCREENING-DISCHARGE ANALYSIS[60]

Height of overflow discharge	8 in.	9½ in.	6 in.
Screening used	12-mesh	12-mesh	¼-in. mesh
Percentage of solution in ore pulp	68.6	84.8	82.0
R.p.m. of mill	7½	7½	8
Screening Analysis			
	Per cent.	Per cent.	Per cent.
+65	30.9	11.9	35.6
+100	15.12	16.2	9.5
+200	19.9	25.6	13.2
−200	34.08	46.3	41.7
	100.00	100.00	100.00

The **Operation of High- and Low-Speed Chilean Mills** is given in Table LVI.

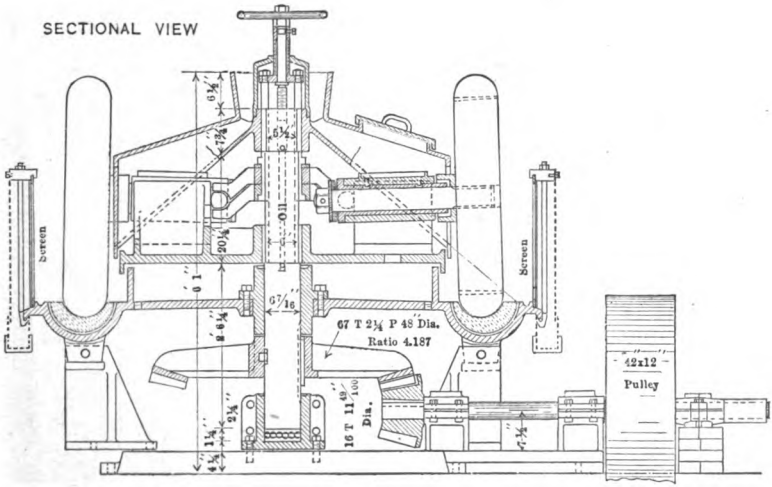


FIG. 104.—Denver quartz mill.[61]

TABLE LVI.—COMPARISON BETWEEN HIGH- AND LOW-SPEED CHILEAN-MILL OPERATION

Mill	Type	Speed, r.p.m.	Capacity, tons per day	Screen mesh	Feed		Product, per cent.											Hp.
					Size	Hardness	+20	+30	+40	+50	+60	+80	+100	+120	+150	+200	−200	
High Speed																		
Portland	Akron, 6 ft.	34	124	30	¼ in.	Hard	..	..	...	..	...	..	..	...	...	..	...	..
Independence	Akron, 6 ft.	33	120	30	⅜ in.	Hard	..	..	...	21	...	..	12	...	5	62	...	55
Goldfield	Trent, 6 ft.	32	75	30	4 mesh	Medium	..	4	...	10	...	16	9	...	6	6	48	75
Mogul	Monadnock	..	105	30	1.5 in.	Hard	..	14.5	...	..	17.8	..	12.6	...	...	15.7	39.4	45
Low Speed																		
Gold Belt	Lane, 7 ft.	18	25,	None	1 in.	Hard	0.1	0.27	0.37	..	1.50	3.87	8.10	...	...	35.79	50.0	10
Sta. Elena	Lane, 7 ft.	15	25	40	2 in.	Hard quartz	..	..	...	..	1.15	5.65	0.15	...	...	25.75	67.3	15
La Union	Lane, 8 ft.	10	20.0	None	1.5 in.	Medium	..	..	...	..	...	0.30	1.50	...	13.45	4.75	80.0	13
El Bote	A.C. 8 ft.	12	18.6	None	2 in.	Hard	..	..	...	..	...	..	...	...	...	30	70.0	13
Defender	Lane, 10 ft.	8	36	None	.....	Medium	..	..	0.54	..	...	5.80	13.70	...	...	30.51	69.5	10
Llano	Lane, 10 ft.	8	48	None	1 in.	Very hard	..	0.42	0.858	..	2.37	5.34	13.84	3.65	...	3.48	70.0	10
Musgrove	Lane, 10 ft.	8	25	None	1½ in.	Extremely hard	..	..	...	..	...	..	...	...	...	30	70.0	10

The **Denver Quartz Mill** (Fig. 104) consists of four rollers with convex edges running in a concave die. A scouring action is claimed, which favors the subsequent amalgamation of gold. The mills are built in three standard sizes. viz.: 3 ft. 6 in., 5 ft. 6 in., and 8 ft. in diameter. The muller rings and mortar liners are made of steel or hard iron.

The **Huntington Mill** consists of a circular pan with a central

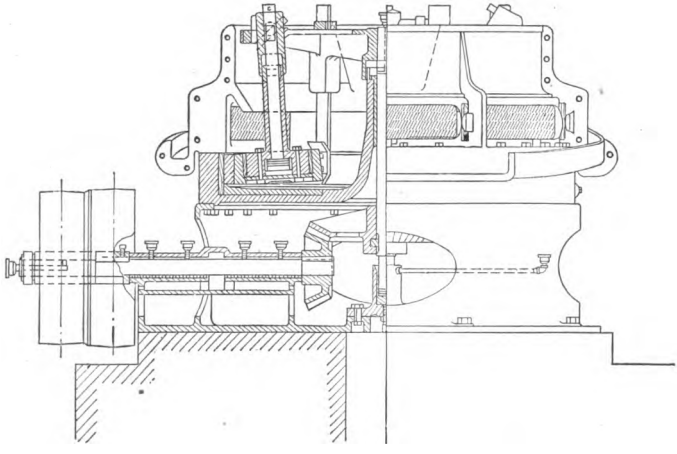


FIG. 105.—Huntington mill with iron base.[8]

spindle from which are suspended four arms carrying rollers. These rollers operate centrifugally and crush the material against a ring die. A screen is arranged just above the level of the rollers. This type of mill has a limited application, on account of the necessity for a smaller feed than is required by many other secondary crushers, but has been used largely as an amalgamator and crusher on small properties. A diagram of this type of mill will be found in Fig. 105 and details of operation in Table LVII.

TABLE LVII.—HUNTINGTON-MILL DATA[8]

Size, ft.	Total weight, lb.	Weight of heaviest piece, lb.	Pulleys, inches	H.p.
3½	6,700	1,600	24 × 5½	5
3½ Sect.	6,900	300	24 × 5½	5
5	16,300	5,500	30 × 8½	8
6	44,000	14,050	48 × 12½	15

Capacity in Tons per 24 Hours and Speed of Countershaft for Different Feeds

Size of mill, ft.	¼-in. feed			½-in. feed			¾-in. feed		
	R.p.m.	Capacity	Mesh	R.p.m.	Capacity	Mesh	R.p.m.	Capacity	Mesh
3½	150	5 to 9 15 to 22 23 to 46	100 40 20	135	15 to 22 26 to 38	40 20			
5	166	9 to 15 25 to 35 50 to 75	100 40 20	150	25 to 35 42 to 62	40 20	134	20 to 28 35 to 50	40 20
6	216	17 to 28 47 to 65 92 to 140	100 40 20	195	47 to 65 78 to 115	40 20	174	38 to 52 65 to 93	40 20

The **Griffin Mill** operates by means of a single roller attached to a vertical shaft which is thrown out by centrifugal force against a ring die. Details of the operation of 16 of these mills, since replaced by Krupp ball mills, are given in Table LVIII. The fineness of the product will be noted. The mill has found small favor among operators, on account of constant necessity for repairs and renewals.

TABLE LVIII.—GRIFFIN-MILL DATA[7]

Mine	Gt. Boulder Perseverance.
Size of mill	No. 11.
Diameter of crushing roll	15 in.
Diameter of ring die	30 in.
R.p.m. of shaft	198
Life of die rings	18 days
Life of roll bodies	6 months
Life of roll tires	9 days
Size of feed	To pass 1¼-in. ring
H.p. consumption	18
Output	37 tons per day
Screening	14 × 14 × 22 I. S. W. G.

Screening analysis

Mesh	With 14-mesh screen of 22 I. S. W. G., per cent.	With 10-mesh screen of 21 I. S. W. G., per cent.	With 10-mesh screen of 20 I. S. W. G., per cent.
+ 20	0.30	3.50	2.50
− 20 + 30	1.00	5.15	4.00
− 30 + 40	2.20	4.75	4.50
− 40 + 60	5.50	8.75	7.15
− 60 + 80	5.35	5.25	5.35
− 80 + 100	4.15	4.00	4.85
− 100 + 120	2.00	2.65	2.25
− 120 + 150	7.35	5.00	6.00
− 150 + 200	4.05	2.15	1.65
− 200	68.10	58.80	61.75
	100.00	100.00	100.00

The **Sturtevant Ring Roll Mill** (Fig. 106) takes a feed up to 1½ in. In the operation of this type of mill the ring is revolved and the rolls are held against the ring by pressure. The product from the mill may be screened and the oversize returned for further reduction (see Fig. 107).

The **Marathon Mill** (Fig. 108) consists of a grinding drum, mounted on a cradle in such a way that fall between feed and discharge levels may be adjusted to suit requirements. The

grinding media consist of loose steel bars of a longitudinal dimension equal to the length of the mill. The shell is lined with steel plate having corrugations which tend to produce a wave

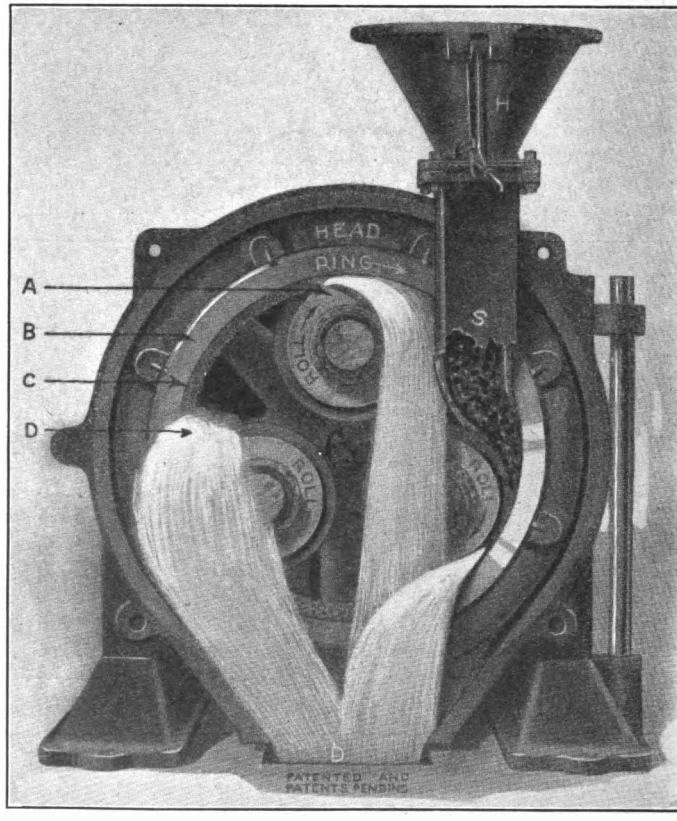


FIG. 106.—Sturtevant ring-roll mill.[32]

Feed enters hopper at *H*.
Spout *S* delivers it at centre of concave revolving ring where it is strongly held by centrifugal force until crushed off by the rolls, discharging at *D*.
A Spring-pressed rolls driven by centrifugally-held material.
B Rigid revolving ring drives all three rolls on 1″ layer of centrifugally-held material.
C Thick layer of centrifugally-held, unground rock.
D Ground rock crushed on both sides of ring.

motion among the rods during the revolution of the mill. The mill is driven by spur gearing which is arranged centrally and

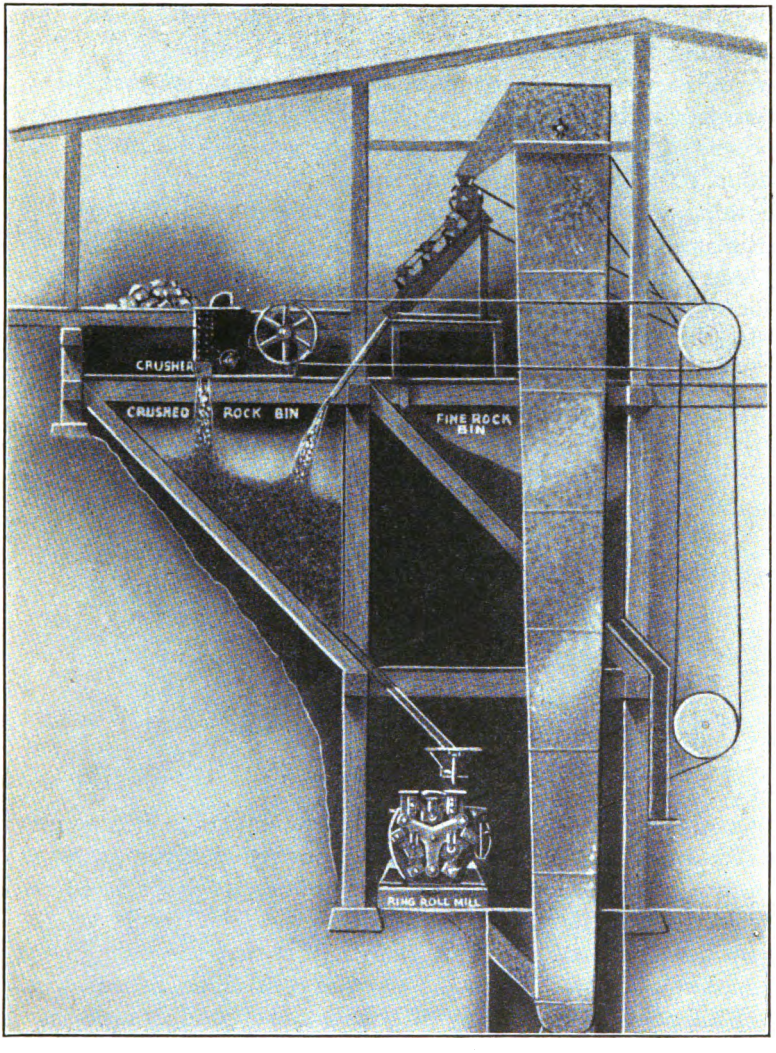


FIG. 107.—Ring-roll grinding with return system.[32]

between two tire rings that rest on four roller-ring bearings. An important feature of the operation of the Marathon mill is that the rods space themselves at definite distances from each other, according to the size of the particles in the feed and the product, the spacing decreasing toward the discharge end of the mill. The action is, therefore, somewhat similar to that occurring in roll crushing, small particles of ore being able to pass through the entire mill without further and possibly uneconomic reduction in size. The efficiency of the mill is, therefore, high—the momentum of each rod (and consequently its effectiveness) is distributed over the entire length. This is in distinct contrast to

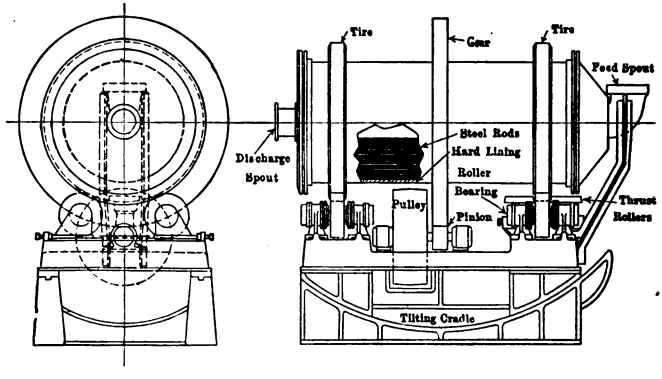


FIG. 108.—Marathon mill.[42]

the action in an ordinary ball mill, where impact is, necessarily, concentrated in one spot, and where low efficiency may result because of unproductive blows—unproductive either from the mechanical or economic point of view. The result of the distributed impact in a Marathon mill is that the discharge is remarkably even. When the various controlling factors have been adjusted, in conformity with the grade of product necessary, the results are often noteworthy. Other than under abnormal conditions, little or no undersize and practically no oversize escapes. The general characteristics of the discharge will alter proportionately as the feed is forced, but the evenness of the

TABLE LIX.—MARATHON-MILL PERFORMANCE [57]

	Test No. 1	Test No. 2
Ore	Quartz monzonite porphyry	
Size of mill	3 ft. by 7 ft.	
Iron rod size	½ in. to 2 in. diam.	
Charge of rods	7000 lb.	
Motor used	25 h.p.	
Speed	30 r.p.m.	
Ratio solids to water	1 to 1.8	1 to 0.57
Horsepower per hour	18.60	22.50
Dry feed per horsepower hour, tons	0.5316	0.8184
Tons dry feed per 24 hr	236.5	440.0
Wear of liners per ton ground	0.2313 lb.	0.14149 lb.
Wear of rods per ton ground	0.63697 lb.	0.39295 lb.

Screening Analysis

Mesh	Opening, in.	Opening, mm.	Test No. 1		Test No. 2	
			Feed, per cent.	Discharge, per cent.	Feed, per cent.	Discharge, per cent.
+ 4	0.185	4.699	0.13	....	0.09	.....
+ 6	0.131	3.327	6.49	....	9.37	0.04
+ 8	0.093	2.362	17.12	0.01	21.58	0.81
+ 10	0.065	1.651	13.40	0.28	15.90	4.90
+ 14	0.046	1.168	10.44	2.12	13.55	12.05
+ 20	0.0328	0.833	8.51	7.06	11.14	15.55
+ 28	0.0232	0.589	8.35	14.77	10.42	16.82
+ 35	0.0164	0.417	6.61	14.43	6.96	12.50
+ 48	0.0116	0.295	5.30	11.68	3.92	8.53
+ 65	0.0082	0.208	3.91	8.72	1.97	5.43
+ 100	0.0058	0.147	3.78	8.05	1.15	4.67
+ 150	0.0041	0.104	2.46	5.15	0.66	2.82
+ 200	0.0029	0.074	1.29	2.67	0.37	1.70
− 200	.....	.....	12.21	25.06	2.92	14.18
			100.00	100.00	100.00	100.00

product will remain a distinguishing feature of the result. The wear on rods is not abnormal, but they must be replaced when worn to a certain size; and this detracts, to a small extent,

from the net efficiency of results. The principle underlying the action of the mill is sound.

Details of Results of Marathon-Mill Operation are given in Table LIX and refer to tests made with this type of mill in competition with Hardinge pebble and Chilean mills. Improved results were obtained in the second test and may be traced to (*a*) better classification of feed and elimination of slime from same, (*b*) higher tonnage, and (*c*) reduction in feed moisture percentage. The reduced wear in rods and liners in the second test is also an indication that the previous test was made on an insufficient feed to protect these parts from self-abrasion. Manufacturer's information about these mills is given in Table LX.

TABLE LX.—MARATHON-MILL DATA [42]

No.	Size		Capacity in tons per hour		Horse power, about	Weight about, lb.
	Outside diameter	Inside length	At initial grinding	Stage reduction or regrinding		
1	16 in.	by 3 ft.	up to ½ or ¾	up to 1 to 1½	1	2,000
2	16 in.	by 4 ft.	up to ¾ or 1	up to 1½ to 2	1	2,500
3	22 in.	by 6 ft.	up to 2 or 3	up to 4 to 6	4	7,000
4	26 in.	by 8 ft.	up to 4 or 5	up to 8 to 10	8	12,000
5	26 in.	by 10 ft.	up to 5 or 6	up to 10 to 12	10	13,000
6	36 in.	by 8 ft.	up to 10 or 12	up to 20 to 25	20	26,000
7	36 in.	by 10 ft.	up to 12 or 15	up to 25 to 30	25	32,000
8	48 in.	by 8 ft.	up to 25 or 30	up to 50 to 60	35	45,000
9	48 in.	by 10 ft.	up to 35 or 40	up to 70 to 80	45	55,000

No.	Floor space, width, length, height, in feet			R.p.m., drum	Pulley size diam. and face, in.	Rod charge, lb.
1	2	4½	3	42	15× 5	850
2	2	5½	3	42	15× 5	1,110
3	2½	7½	4½	38	24× 6	2,160
4	4	11	7	35	24×10	3,500
5	4	13	7	35	24×10	4,000
6	5½	11	9	30	29×11	7,000
7	5½	13	9	30	29×11	9,000
8	7	11	10	28	38×12	14,000
9	7	13	10	28	38×16	18,000

SECTION XI

ROLL CRUSHING

Crushing Rolls (Fig. 109) consist, fundamentally, of two parallel shafts, one of which is fixed and the other movable. These shafts carry metal cores on which are fixed the crushing shells. These latter consist of short cylinders of steel and

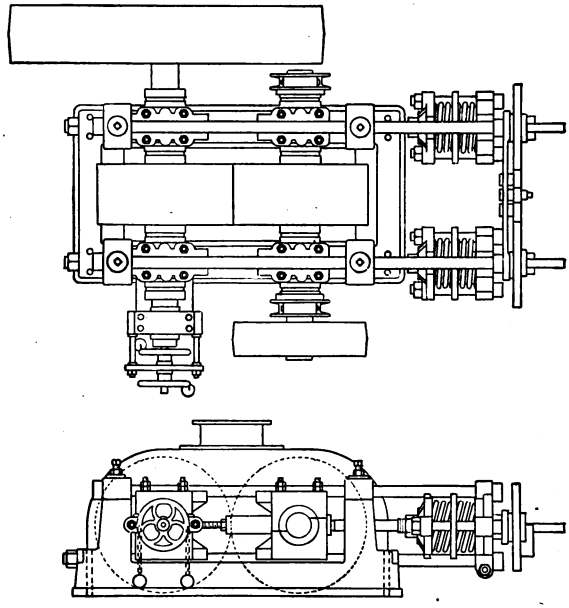


FIG. 109.—Crushing rolls.[31]

revolve in opposite directions, tending to lock the ore fed between them and by a pinching action crush it to a size proportionate to the dimension of the opening between roll faces. Crushing pressure is maintained by means of springs.

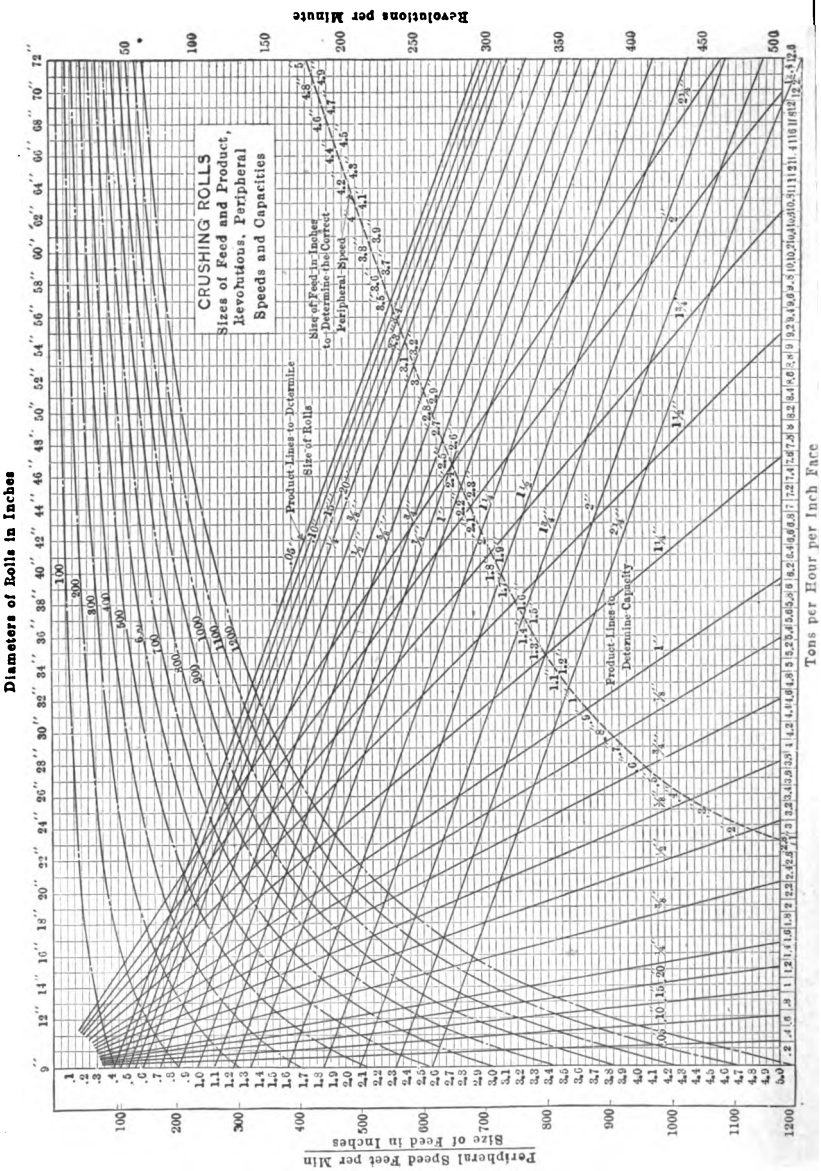
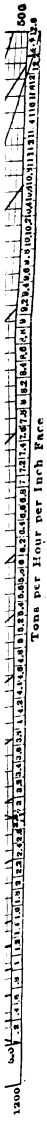

CRUSHING ROLLS
Sizes of Feed and Product, Revolutions, Peripheral Speeds and Capacities
Diameters of Rolls in Inches
Revolutions per Minute
Peripheral Speed Feet per Min
Size of Feed in Inches
Tons per Hour per Inch Face
Product Lines to Determine Size of Rolls
Size of Feed in Inches to Determine the Correct Peripheral Speed
Product Lines to Determine Capacity

FIG. 110.—Crushing-roll diagram.[11]



The **Crushing-Roll Diagram** (Fig. 110) may be used for the determination of average roll sizes, capacities, and speeds. An illustration of the use of the diagram on the assumption that it is required to reduce 25 tons per hour of 2-in. material to ½ in. is given as follows:

To determine diameter of roll required: Find the size of feed (2 in.) in the column at the left of the diagram and follow the horizontal line leading from this size to the point of intersection with the diagonal ½-in. product line in the series marked "Product lines to determine size of rolls." From this point follow the nearest vertical line to the top of the diagram. The roll diameter thus found (38 in.) is the theoretical minimum diameter that can be used to reduce 2-in. feed to ½ in. As this is not a standard diameter, the nearest larger size of standard dimension should be used, which would be 42 in.

To determine speed at which 42-in. rolls should run: Find the size of feed on the curve marked "Size of feed in inches to determine correct peripheral speed," and follow the horizontal line to the left, to the column marked "Peripheral speeds, feet per minute," which will give a speed of 700 ft. per minute.

To determine the capacity of a 42-in. roll running at 700 ft. per minute, making a ½-in. product: From the given speed (700 ft. per minute) follow the horizontal line to the right to the point of intersection with the ½-in. product line in the series marked "Product lines to determine capacities." Directly underneath this point at the bottom of the table will be found a capacity given of 1.8 tons per hour per inch of face. From this we find that it will require a roll 14 in. wide to give a capacity of 25 tons per hour, but as this is not a standard width for a 42-in. roll, a 42 by 16-in. roll may be adopted.

Summing up the above we have determined that a 42 by 16-in. roll running at a speed of 700 ft. per minute will reduce 28.8 tons of 2-in. material to ½ in. per hour.

The above capacities are based on 25 per cent. of the theoretical tonnage of a ribbon of crushed ore of thickness equal to the distance between the roll faces, and assuming that the ore

weighs 100 lb. per cubic foot. In the above example the theoretical capacity in tons per hour

$= (60 \times 16 \times \frac{1}{2} \times 200 \times 100) \div (2000 \times 144)$
$= 116$ tons per hour
$= 29$ tons per hour at 25 per cent. efficiency.

To find the R.p.m. of the roll at the above peripheral speed: Follow the curve for 700 ft. per minute to the point where it intersects the vertical line under 42-in. diameter roll, and then follow the horizontal line from this point to the right of the diagram, where it will be found that this corresponds to 63 r.p.m.

Crushing-Roll Capacities at Various Speeds may be determined by the aid of Table LXI, giving certain factors in ratio to peripheral speed in feet per minute. The capacities resulting are 25 per cent. of the theoretical capacity and the weight of ore has been assumed to be 100 lb. per cubic foot. To find the capacity of the rolls in tons per hour, multiply the factor opposite the peripheral speed by the distance the roll faces are set apart in inches, and the result by the width of the roll face in inches.

TABLE LXI.—FACTORS FOR THE DETERMINATION OF CRUSHING-ROLL CAPACITIES AT VARIOUS SPEEDS [11]

Peripheral speed, ft. per min.	Factor	Peripheral speed, ft. per min.	Factor	Peripheral speed, ft. per min.	Factor
400	2.08	675	3.51	950	4.94
425	2.21	700	3.64	975	5.07
450	2.34	725	3.77	1000	5.20
475	2.47	750	3.90	1050	5.46
500	2.60	775	4.03	1100	5.72
525	2.73	800	4.16	1150	5.98
550	2.86	825	4.29	1200	6.24
575	2.99	850	4.42	1250	6.50
600	3.12	875	4.55	1300	6.76
625	3.25	900	4.68	1350	7.02
650	3.38	925	4.81	1400	7.28

Dry-Crushing Rolls, although possessed of a wide range of adaptability, have an application limited to the production of material usually no finer than 10 to 15 mesh. **Wet-Crushing**

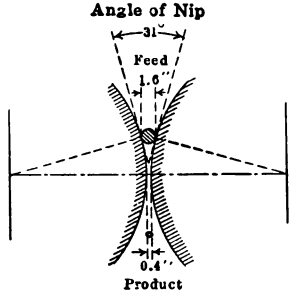


FIG. 111.—Diagram illustrating angle of nip in roll operation.[16]

Rolls may be operated to produce a finer product, *e.g.*, 30 mesh. Best practice in either case is indicated where a reduction of no more than four to one in diameter is achieved in one passage through each machine.

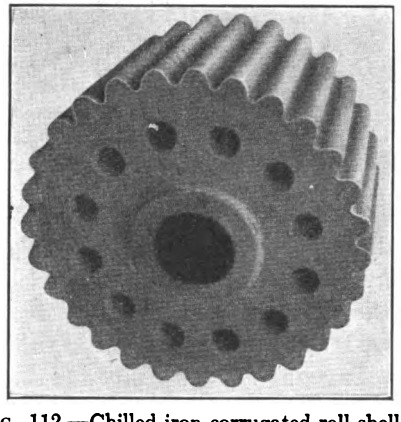


FIG. 112.—Chilled-iron corrugated roll shell.[18]

The **Angle of Nip** (Fig. 111) is the angle formed by tangents from the points of contact of rock and roll periphery. This angle varies with the diameter of the rolls and the distance

between the faces. The coefficient of friction is usually taken as 0.30 and the angle of nip 31°.

Corrugated Roll Shells (Fig. 112) enable a larger-sized feed to be handled in a small set of rolls than would be the case if smooth-faced shells were used.

Even **Wear of Tires** is aided by an adjustment arranged to move the stationary roll in either direction along its axis. A type of this apparatus is illustrated in Fig. 113.

Hollow **Roll Faces** are usually caused by uneven feeding.

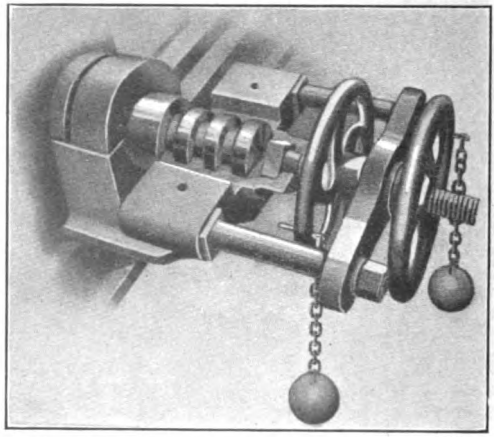


FIG. 113.—Lateral adjustment mechanism.[31]

This necessitates the removal of the rolls and the turning down of the faces in a lathe.

Corrugation of Roll Faces is largely caused by excessive size of feed beyond the dimension indicated by the angle of nip as the maximum. A second cause is found in the case where both roll faces are not traveling at identically the same speed.

Rolls with Fleeting Action (Fig. 114) carry an automatic lateral adjusting device whereby one roll moves across the face of the other roll twice an hour. The degree of movement depends in some measure on the size of the material being crushed. The object of this device is to prevent corrugations.

Adjustment of **Product Size** is usually made by means of gears operated by a crank. In the type illustrated (Fig. 115) parallelism of the roll faces is insured by simultaneous movement of both roll faces, and the desired adjustment is maintained by means of

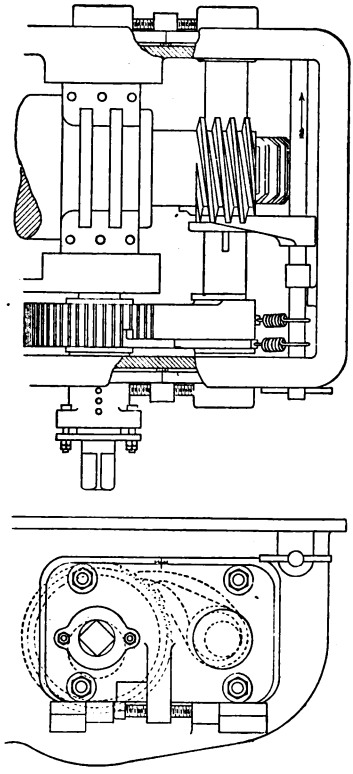


FIG. 114.—Fleeting roll mechanism.[18]

a key engaging the teeth of the pinion to which the shaft is attached.

Springs in **Rolls** are provided in order to prevent damage to frames and tension rods by the presence of steel or other unbreakable material in the feed.

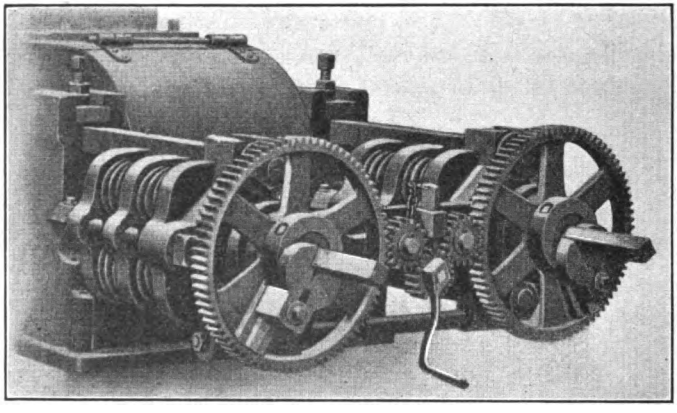


FIG 115.—Roll adjustment gearing.[31]

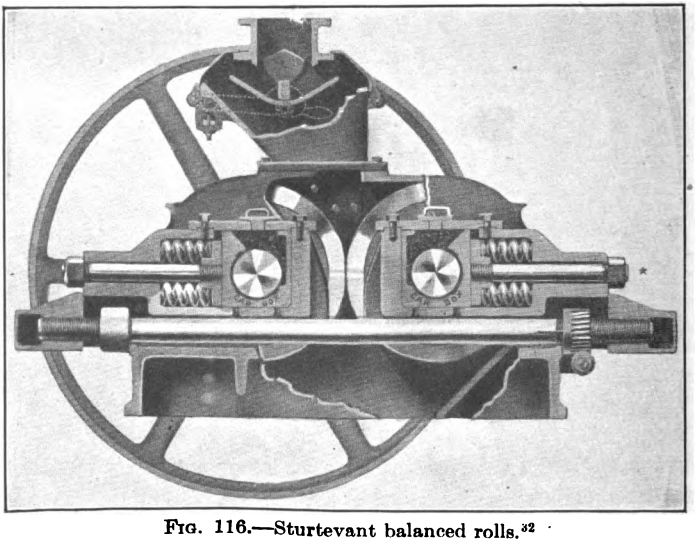


FIG. 116.—Sturtevant balanced rolls.[32]

Rigid Rolls, as their name implies, have no springs and, in consequence, operate with less vibration. They are particularly well adapted for crushing comparatively fine product where there would be less likelihood of the presence of steel of a size to do material damage.

Balanced Rolls (Fig. 116) are designed so that parts moving oppositely have equal weights and meet at equal speeds, and it

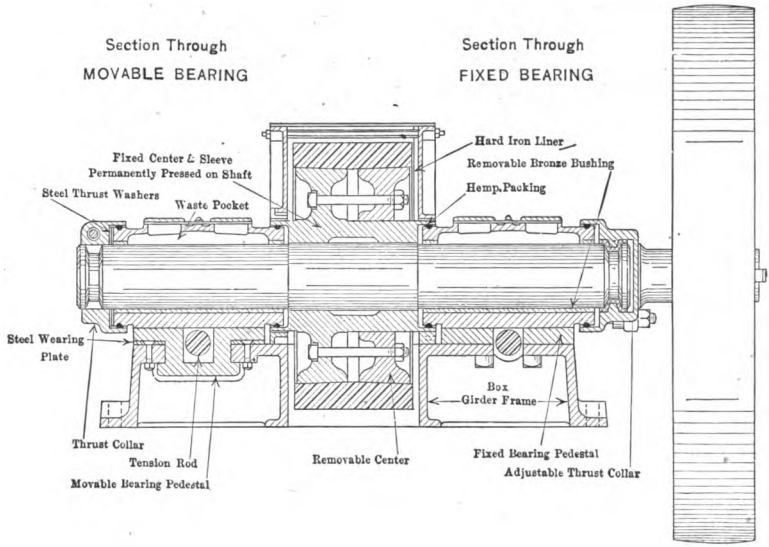


FIG. 117.—Convertible-type crushing rolls.[16]

is claimed that shocks are balanced, stresses minimized, and corrugations avoided to a considerable degree.

In **Convertible-Type Rolls** the rigid and spring roll parts are interchangeable. Details of construction of bearings, frame, and shaft of this type are given in Fig. 117.

Power for Roll Crushing is usually transmitted to a pulley fixed on a shaft extension of the fixed roll. The movable roll is generally connected with a small pulley, the function of which

is to transmit sufficient power to the roll to which it is connected to keep it revolving when not crushing.

The High Pressure Borne by Roll Bearings necessitates ample

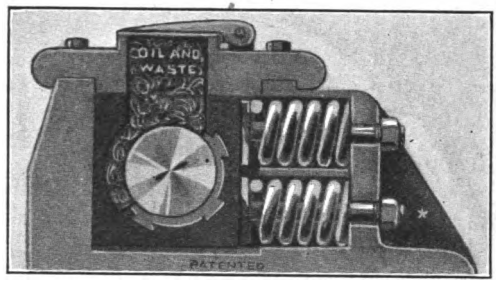


FIG. 118.—Car-box roll bearing.[32]

and efficient lubrication to obviate heating. Figure 118 shows a modification of the ordinary railroad-car journal method of lubrication, which has been successfully adopted for roll requirements.

SECTION XII

REGRINDING OR SLIMING IN PANS AND TUBE MILLS

The Grinding Pan (Fig. 119) consists typically of a circular cast-iron trough, usually 5 ft. in diameter, fitted with flat-faced renewable dies. These form the lower grinding surface, which is stationary. The upper grinding surface consists of renewable shoes attached to a muller plate, spider, yoke, and central shaft,

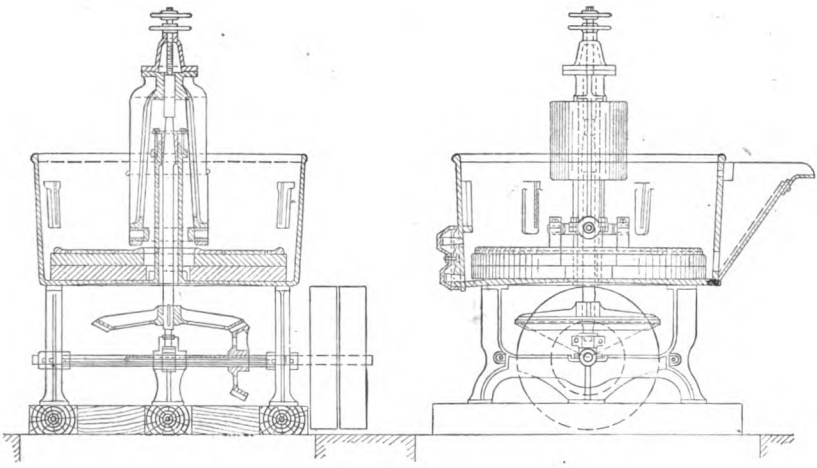


FIG. 119.—Wheeler grinding pan.[7]

to which power is applied through bevel gearing underneath the pan. The discharge is usually in spitzkasten form, and arranged so that any unground product overflowing is returned automatically to the pan for further reduction.

Pan Shoes and Dies vary in number from 8 to 18 of each per pan. They are usually 2½ or 3 in. thick and are made either locally at the mine from pig iron and scrap or other mixture, or

purchased in the form of hematite iron. The life of these wearing parts varies with the kind of work the pan is called upon to perform and the class of material used. Shoes locally cast from scrap usually last about a month. Better-quality material will insure a life of from 6 to 9 months.

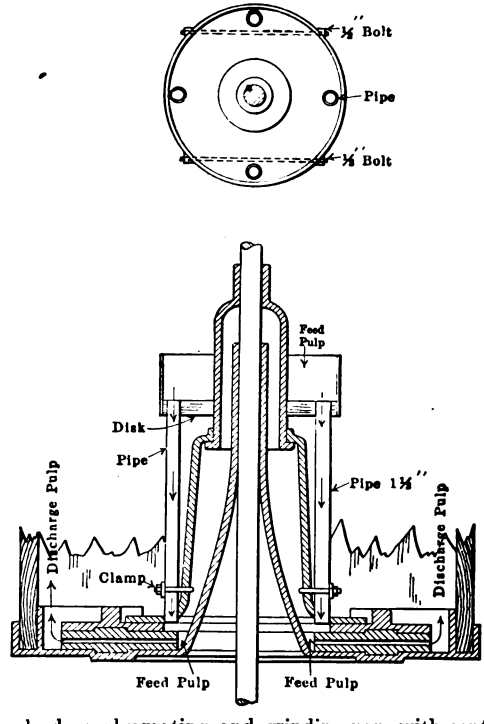


FIG. 120.—Standard amalgamating and grinding pan, with central-pipe feed.[62]

Central Feed for Grinding Pans may be arranged by a system of pipes leading from a feed-box around the top of the spider (Fig. 120) or by means of a central feed cylinder (Fig. 121). With the latter arrangement all feed must necessarily pass between shoes and dies before reaching the pulp-discharge point.

A Classifying Pipe Discharge may be arranged through the side of the pan (see Fig. 121). A pipe extends into the pan

FIG. 87.—Metal stamp-stem guides.[16]

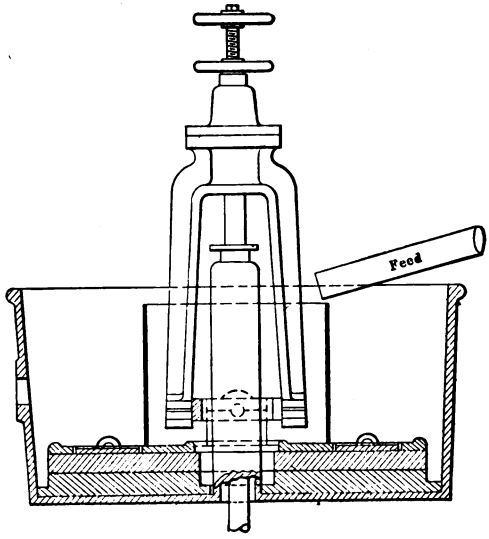


FIG. 121.—Cylinder central feed for grinding pans.

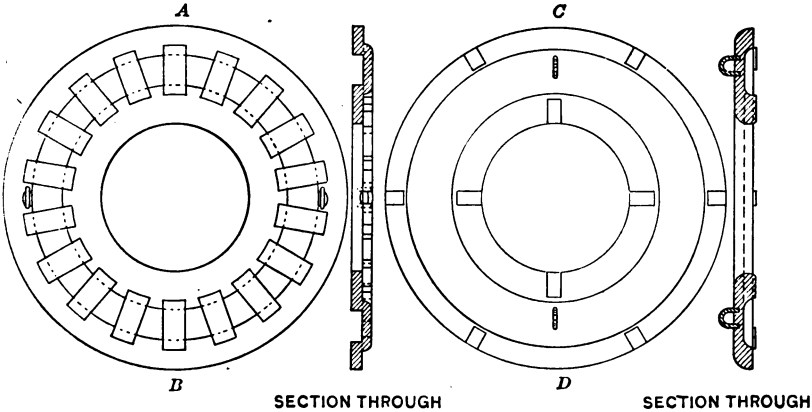


FIG. 122.—Compensating weights for grinding pan.[7]

and terminates about halfway toward the center in a bend or elbow. A classifying action based on centrifugal forces occurs, the coarser particles being kept toward the edge of the pan.

Compensating Weights, to allow for loss of weight occasioned in wear of shoes, are an important factor in the efficient working of an ordinary grinding pan. Inability to realize this fact constitutes one of the principal reasons for failure. The type of weight shown in Fig. 122 is used extensively, and was originally designed at the Ivanhoe Mine, West Australia. The ring weighs 600 lb. and is slipped on over the yoke, and rests on the muller plate. Grading analyses with pans driven at 57 r.p.m. and consuming 6½ h.p. are given in Table LXII.

TABLE LXII.—EFFECT OF COMPENSATING WEIGHTS IN PAN GRINDING[7]

Discharge grading analysis

Screening	18 tons per day		22 tons per day	
	With compensating weights, per cent.	Without compensating weights, per cent.	With compensating weights, per cent.	Without compensating weights, per cent.
+ 40	.....	.....	0.1	0.5
− 40+ 60	0.3	3.5	1.7	16.7
− 60+100	11.7	48.5	24.2	50.2
−100+150	11.7	10.2	18.5	8.9
−150	76.3	37.8	55.5	23.7
	100.0	100.0	100.0	100.0

The **Advantages of Grinding Pans** for small- or medium-tonnage operations are seen in

(i) Immediate accessibility of all parts.
(ii) Availability as amalgamators.
(iii) Small unit capacity, resulting in less disorganization in mill output during repair.
(iv) Low unitary power consumption with no extra or overload provision necessary for starting up.
(v) Wearing parts of simple shape and design and can be cast from scrap metal.
(vi) Rapid discharge of ground material with less necessity for elimination of fines from feed than with other types of fine grinders.
(vii) Automatic return of unground product to body of pan.

Criticism Against Grinding Pans has been too often based on the experience with an ancient type of apparatus known as the standard grinding and amalgamating pan (Fig. 120). In this machine the number and dimensions of both shoes and dies are inadequate, the grinding area is insufficient, and no provision exists to maintain grinding-surface weight. The principal objection against grinding pans is that they require adjustment and are not so fool-proof as tube mills.

Grinding Pans may be used either as **Stage Grinders** or as **Slimers.** Details of operation in both capacities are given in Tables LXIII, LXIV, and LXV.

TABLE LXIII.—OPERATION OF PANS USED FOR STAGE GRINDING[7]

Mine	Sons of Gwalia	L. V. Consols	Gt. Fingall
Material	Classified sand product	Middlings from Wilfleys	Sand
Condition of material	Raw	Raw	[Raw
R.p.m.	45	45	45
H.p.	5⅓	7	5
Size of pan	5 ft.	5 ft.	5 ft.
Tons per day	33	30	33
Type of pan	Wheeler	Wheeler	Forwood Down

Screening Analysis

Mine	Sons of Gwalia		L. V. Consols		Gt. Fingall	
Mesh	Feed, per cent.	Discharge, per cent.	Feed, per cent.	Discharge, per cent.	Feed, per cent.	Discharge, per cent.
+ 40	50.5	12.5	28.34	4.00	66.8	26.6
− 40 + 60	30.0	27.5	9.19	2.5	.....	.....
− 60 + 80	.....	.....	21.06	14.75	.....	.....
− 80 + 100	17.5	24.0	6.32	8.52	.....	.....
− 100 + 120	.....	.....	9.88	17.80	.....	.....
− 120 + 150	1.5	7.5	2.78	5.65	18.2	50.3
− 150	0.5	28.5	22.43	46.78	15.0	23.1
	100.0	100.0	100.00	100.00	100.0	100.0

TABLE LXIV.—OPERATION OF PANS USED FOR FINE GRINDING[7]

Mine	Kalgurli	Associated	L. V. Consols
Material	Sand	Ore	Concentrate
Condition	Roasted	Roasted	Roasted
Gold recovered in pans by amalgamation	12 % of value of ore	22 % of total recovery	51 % of total value of concentrate
R.p.m	53	47	40
H.p.	.....	5.2	7.0
Size of pan	5 ft.	5 ft.	5 ft.
Tons per day	8–10	15	30
Type of pan	Wheeler	Wheeler	Wheeler

Screening Analysis

Mine	Kalgurli		Associated		L. V. Consols	
Mesh	Feed, per cent.	Discharge, per cent.	Feed, per cent.	Discharge, per cent.	Feed, per cent.	Discharge, per cent.
+ 40	18.1	.....	5.99	0.05	17.81	.....
− 40 + 60	21.2	.....	16.45	0.08	5.87	0.38
− 60 + 80	....	.....	10.90	0.28	19.52	3.97
− 80 + 100	42.2	.....	7.63	0.89	5.55	4.35
− 100 + 120	6.1	1.16	4.31	1.63	9.07	9.30
− 120 + 150	6.1	5.89	4.95	2.51	4.77	5.60
− 150	6.3	92.95	49.77	94.56	37.41	76.40
	100.0	100.00	100.00	100.00	100.00	100.00

Wheeler-Type Grinding Pans were previously in use at the Homestake. These were 5 ft. in diameter, consumed 8.4 h.p. at 58 r.p.m. and handled just under 20 tons per day per pan. Grading analyses of feed and discharge will be found in Table LXVI.

Grinding-Pan Practice with Closed Circuit and Dorr Classification is in operation in Bolivia, and, according to Söhnlein, a pan, with a central feed of 38 tons per day, running at 60 r.p.m. with a consumption of 6.26 h.p., is capable of grinding 30 tons of sand per day to pass 200 mesh. The feed showed about 70 per cent. between 30 and 100 mesh.

TABLE LXV.—AUSTRALIAN GRINDING-PAN PRACTICE[63]

Data	Perseverance	South Kalgurli	Great Boulder Prop.	Assoc. Northern
Monthly tonnage	19,000	9,030	18,000	3,700
Daily tonnage	633	301	600	123
Tons on 150 mesh (requires grinding)	209	131	225	56
Tons through 150 mesh (already ground)	424	170	375	67
On 150, per cent	33	43.6	37.5	45.5
Through 150, per cent	67	56.4	62.5	54.5
After grinding in pans, per cent. on 150	28.4	12.6	29.0	12.5
After grinding in pans, per cent. through 150	71.6	87.4	71.0	87.5
Number of pans	17	7	22	8
Diameter of pans	8′	Five 8′ and Two 5′	5′	5′
Revolutions per minute	10–30 and 7–25	33 and 55	60	47
Total die area of pans in square feet	537	183	279	101
Horsepower per pan	8.0	12 and 7	7	5.1

In the **Cobbe Pan** the pressure between shoes and dies is maintained constant by the use of levers that are weighted in such a way as to keep the dies in adjusted pressure against the shoes. This type of pan is arranged for screen discharge, is fitted with arrangements for central feed, and is particularly well adapted for stage grinding. Details of operation are given in Table LXVII.

TABLE LXVI.—HOMESTAKE PAN GRADING ANALYSES[51]

	Feed, per cent.	Discharge, per cent.
+ 50	38.4	4.0
+ 80	42.0	12.5
+ 100	9.9	13.5
+ 200	6.8	28.0
− 200	2.9	42.0
	100.0	100.0

TABLE LXVII.—COBBE GRINDING-PAN DATA

Screening on battery..... 8 × 8 mesh
Screening on pan..... 14 × 14 mesh
Feed to pan..... 66.6 tons per 24 hr.
R.p.m. of pan..... 44

Grading Analysis

Mesh	Feed, per cent.	Discharge, per cent.
+ 20	17.70	0.03
+ 40	16.73	1.43
+ 60	9.30	10.93
+ 80	7.40	14.43
+ 100	5.60	10.17
+ 120	1.63	2.17
+ 150	3.06	8.27
− 150	38.58	52.57
	100.00	100.00

Tube-Milling Practice illustrates an important adaptation in ore-reduction methods. Originally borrowed from the cement industry, and worked without modification in design, the machine has passed through many stages of improvement. The original silex lining of flint blocks has been largely replaced by hard

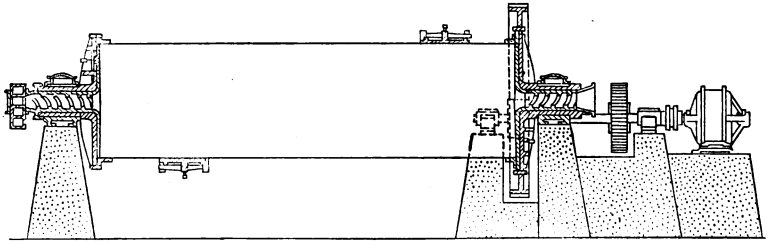


FIG. 123.—Tube mill with motor drive.[18]

metal lining. The expensive imported flints used as grinding media are rarely available, and manufactured pebbles of hard rock, mine rock, or small steel balls are generally used.

Tube Mills are usually constructed of a diameter varying between 4 and 6 ft. The length depends on practice requirements, such as fineness of grinding desired and tonnage to be handled. They are arranged to operate horizontally and without fall. Power is transmitted through suitable gearing, and the mill is usually thrown into or out of action by means of some type of friction clutch, which should be of ample capacity (usually twice the average mill load). The initial power required to start a mill of this type is high, especially where the contents have cemented to a certain extent since the previous shut-down.

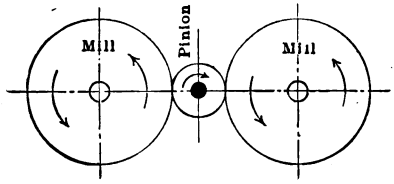


FIG. 124.—Floating tube-mill pinion.

Power for Tube Mills may be transmitted by belting or chain drive; or the countershaft may be motor connected by suitable flexible coupling (Fig. 123). A floating pinion (Fig. 124), with balanced thrusts, may be arranged by driving two mills,

placed parallel, from the one pinion countershaft. By arranging a wide sole plate for the trunnion bearings, fitted with adjusting screws, either mill may be thrown out of action for repair or examination by unclutching the pinion shaft, and moving the mill on the sole plates so that the spur-wheel teeth are out of range of the driving pinion.

Tire and Roller-Bearing Tube Mills (Fig. 125) are said to consume less power than those equipped with trunnion bearings. A combination of trunnion and tire-and-roll bearings is sometimes used. The principal advantage of tire-and-roller support is seen

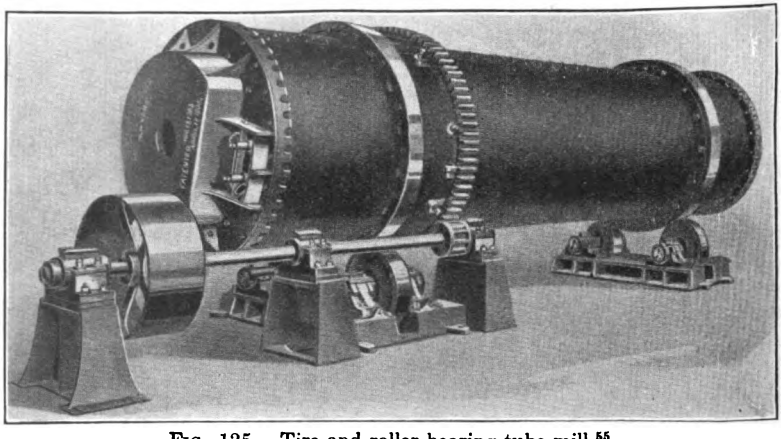


FIG. 125.—Tire and roller bearing tube mill.[55]

in increased diameter of discharge opening screen grizzly (Krupp type), or large diameter spiral discharge (Abbé type).

Gear Drives for **Tube Mills** are shown in Figs. 126 and 127. The spur-gear drive is usually adopted, results in lower horsepower consumption, and is an arrangement permitting greater rigidity in countershaft operation.

Tube Mill Driving Shaft should be as near in line with the horizontal axis of the mill as possible, so that a single sole plate may be used for trunnion and countershaft bearings, with the advantage of greater rigidity and easier adjustment. The wear of white metal used in babbitting the trunnion bearings causes a

sinking of the mill. This necessitates an adjustment of the countershaft and pinion if the latter are not placed as suggested.

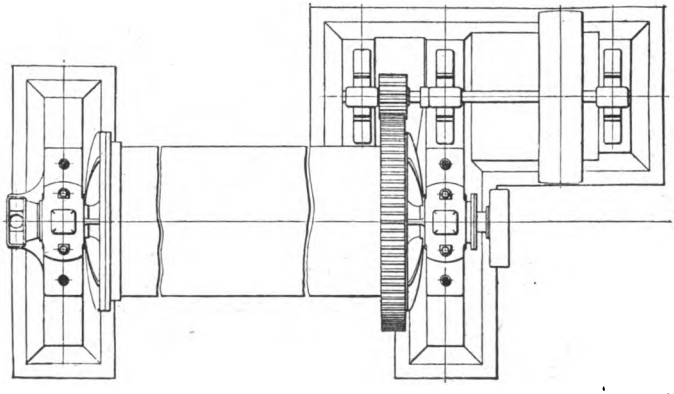


FIG. 126.—Spur gear tube-mill drive.[11]

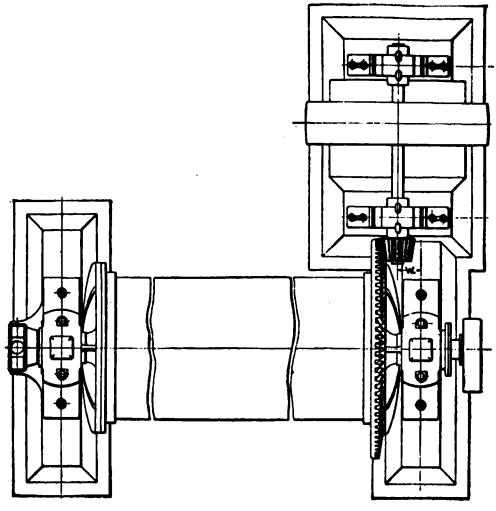


FIG. 127.—Bevel gear tube-mill drive.[11]

Roller Bearings for tube-mill trunnion bearings have been tried, and their adoption has resulted in a marked decrease in power

consumption. Conditions around a tube mill are, however, against keeping the rollers clean, with the result that they have not been adopted to any extent.

Feeding into the Tube Mill is usually accomplished by means of a simple form of scoop spiral, of a size in proportion to the maximum pebble diameter. The feed-end trunnion is lined with a spiral shell, which tends to carry the material into the mill (Fig. 123). The discharge end is sometimes fitted with a reverse spiral for the purpose of retaining pebbles in the mill.

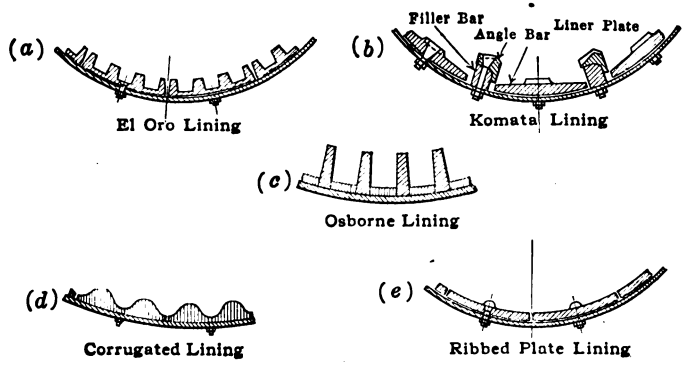


FIG. 128.—Sectional views of tube-mill linings.

Short Tube Mills are used where sliming is not required, and are suitable for regrinding tailings or middlings in concentrator mills, or for the reduction to a suitable mesh of a coarse battery product for subsequent leaching treatment. Details of this class of mill are given in Table LXVIII.

The **Improved Efficiency** resulting from the use of **short tube mills** is receiving increasing recognition. A statement of the results of comparative tests made with reference to the operation of cylindrical and conical mills is given in Table LXIX.

The **El Oro Tube-Mill Lining** (Fig. 128) consists of hard-metal castings carrying grooves having a slight taper inward toward

the mill circumference. These recesses extend along the length of the mill and are designed of a width to allow the pebbles used for grinding to wedge into place, thus forming a protective layer against the metal. This type of lining originated in Mexico, and has been extensively adopted elsewhere.

Corrugated Tube-Mill Lining (Fig. 128) insures considerable movement of the pebbles and prevents slippage and consequent low crushing efficiency. The type has been used largely with Krupp mills.

TABLE LXVIII.—DETAILS OF SHORT TUBE MILLS [18]

Size	Weight, lb.	Weight of iron lining, lb.	H.p.	R.p.m. mill	Charge of flint pebbles, lb.
4 feet 0 inches × 6 feet	14,000	5,400	10	32	3,600
4 feet 6 inches × 6 feet	18,000	6,000	15	29	4,600
5 feet 0 inches × 6 feet	23,000	6,600	23	27	6,000
5 feet 6 inches × 6 feet	30,000	7,200	30	23	7,500
6 feet 0 inches × 6 feet	33,000	8,400	37	21	9,000
7 feet 0 inches × 6 feet	37,000	9,000	45	18	13,000

The **Komata Tube-Mill Lining** (Fig. 128) consists of longitudinal plates, with maximum thickness where wear is greatest, and longitudinal shoes, both of hard steel. The latter serve to raise the pebbles and are the principal grinding-efficiency factor in this type of lining. Each 4-ft.-diameter mill has eight ribs, measuring 3 in. by 2½ in., and the correct speed is given as 27½ r.p.m. For the 5-ft. mill, ten ribs are required, measuring 3½ in. by 3 in., and the mill should revolve at 24 r.p.m. Calculated speed for a mill with Komata lining is considerably below the requirements for ordinary practice.

Ribbed-Plate Tube-Mill Lining (Fig. 128) is a simple type in which a variation in the projection of the ribs may be adopted to modify or increase the degree of lifting of the pebbles.

The **Osborne Tube-Mill Lining** (Fig. 128) consists of steel bars placed alternately lengthwise and edgewise against the inside of the mill shell. The lining is held in position by the wedging

TABLE LXIX.—DETAILS OF SHORT TUBE-MILL OPERATION

7′ × 10′ Power & Mining Cylindrical Tube Mill
vs.
8′ × 30″ Hardinge Conical Pebble Mill

NOTE:—The object in view being to grind the largest possible amount of the material so that it will pass a 40-mesh screen and at the same time avoid producing an excess of 240-mesh material, both mills were in a closed circuit containing a hydraulic classifier and concentrating table so that the oversize of 40 mesh was returned to the tube mills.

	P. & M.	Hardinge
Dimensions of Mills	7′×10′	8′ × 30′
Drives	Individual Motors Belted to Pinion Shaft.	
Gear	100 Teeth	118 Teeth
	4″ Pitch	2½″ Pitch
	127.3″ P.D.	93.9″ P.D.
	14″ Face	6″ Face
Pinion	16 Teeth	17 Teeth
	20.4″ P.D.	13.5″ P.D.
	14″ Face	6½″ Face
Gear and Pinion Ratio	6.25:1	6.94:1
Feed Scoop	3 Cup Spiral	Single Cup Spiral
Dimensions of Trunnion Bearings	24″ × 24″	12¾″ Dia. × 9½″
Wet tons feed per 24 hr	485.00	218.0
Average moisture in feed—per cent	35.9	35.6
Dry tons of feed per 24 hr	310.7	139.8
Ratio of feed to mills	2.22 to 1.00	
Per cent. of material on 40 mesh in feed	83.1	82.9
Per cent. material on 40 mesh in discharge	26.2	26.6
Ratio of reduction on 40 mesh material	3.18:1	3.12:1
Tons reduced through 40 mesh per 24 hr	176.7	78.8
Ratio between mills	2.24 to 1.00	
Per cent. of mineral on 40 mesh in feed	77.3	80.2
Per cent. of mineral on 40 mesh in discharge	18.3	20.9
Ratio of reduction on 40 mesh mineral	4.23:1	3.84:1
Per cent. of material on 80 mesh in feed	98.4	98.4
Per cent. of material on 80 mesh in discharge	56.2	56.7
Ratio of reduction on 80 mesh material	1.75:1	1.73:1
Tons reduced through 80 mesh per 24 hr	131.1	58.3
Ratio between mills	2.24 to 1.00	
Per cent. of mineral on 80 mesh in feed	96.8	97.0
Per cent. of mineral on 80 mesh in discharge	46.2	50.4
Ratio of reduction on 80 mesh mineral	2.09:1	1.92:1
Per cent. of through 240 mesh material in feed	0.1	0.3
Per cent. of through 240 mesh material in discharge	18.0	20.4
Per cent. increase in through 240 mesh material	17.9	20.1
Tons through 240 mesh material produced per 24 hr	55.6	28.1

TABLE LXIX.—*Continued*

	P. & M.	Hardinge
Ratio between mills	1.98 to 1.00	
Per cent. of through 240 mesh material produced in tonnage reduced through 40 mesh	31.5	35.6
Tons reduced through 40 mesh while producing one ton of through 240 mesh material	3.18	2.80
Speeds, R.P.M.	22	30
Speeds feet per minute at large diameter	483	753
Average height of pebble load	Two inches above center.	
Original charge of No. 4 Danish pebbles	20,250 lb.	11,200 lb.
Average pebble consumption per day	979 lb.	521 lb.
Ratio between mills	1.88 to 1.00	
Pebbles consumed per ton material ground	3.15 lb.	3.73 lb.
Pebbles consumed per ton of material reduced through 40 mesh	5.54 lb.	6.61 lb.
Cost of pebbles per ton of material ground	$0.048	$0.057
Cost of pebbles per ton of material ground reduced through 40 mesh	$0.085	$0.101
Average power consumption E.H.P.	91.9	64.4
Power consumed per ton of material ground	7.10 E.H.P.	11.06 E.H.P.
Cost of power per ton of material ground	$0.037	$0.058
Power consumed per ton material reduced through 40 mesh E. H. Powers Hours	12.48	19.61
Cost of power per ton material reduced through 40 mesh	$0.066	$0.103
Tons reduced through 40 mesh per horsepower day	1.92	1.23

action, and no bolts are used. The recessed bars, which are subjected to practically no wear, and consequently do not require renewal, are usually 2½ in. by ¾ in. The radial bars have a slight taper and measure 4 in. by 1¼ in. by ¾ in. This type of lining was designed in South Africa and has been adopted there to some extent.

A **Combination Lining of Cast-Iron Ribs** and **Silex Blocks** is used at the Liberty Bell Mill, according to A. J. Weinig, and it has been found that the addition of the iron rib to every four rows of blocks has resulted in extending the life of the lining to nearly three years of continuous wear. The ribs used are tapered and measure 1¾ × 2 × 4½ in. and are 47½ in. long. The blocks are set with portland cement in sections and steamed for eight hours, the completed lining being finally steamed for 12 hours before the mill is put into commission.

TABLE LXX.—EFFECT OF PEBBLE LOAD VARIATION IN TUBE MILLING[11]

Size of mill, 23 ft. × 5 ft.

Tons ore per 24 hours, 172

Ore, per cent.	63.72	70.17	74.29	60.00	65.38	66.67	66.67	66.67	71.88	71.88	71.88	70.37	70.96	68.18	66.67	70.00	68.00
Solution, per cent.	36.28	29.83	25.71	40.00	34.62	33.33	33.33	33.33	28.12	28.12	28.12	29.63	29.04	31.82	33.33	30.00	32.00
Indicated horsepower	18.80	20.37	22.50	32.16	39.13	42.88	47.16	51.45	56.28	60.10	65.39	77.18	68.61	69.68	75.04	68.60	64.85
Pebble load, tons	1½	3	4½	6	7½	8½	9	9½	10	10½	11	11½	12	12¼	12½	13	13½

Mesh	Feed, per cent.	Grading Analysis, Discharge															
		Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
+ 20	6.0																
+ 30	20.0																
+ 40	24.0																
+ 60	23.0	42.5	46.5	42.0	32.0	29.0	18.0	3.5	4.0	9.0	6.0	6.0	6.0	3.0	4.0	3.0	5.0	8.0
+ 80	11.0																
+100	8.0	27.5	23.5	26.0	32.0	30.0	36.0	29.0	28.0	32.0	30.0	29.0	30.0	27.0	26.0	26.0	28.0	33.0
+120	4.0																
+150	2.0	8.0	8.0	8.0	12.0	14.0	12.0	16.0	13.0	15.0	13.5	15.0	14.0	16.0	13.0	14.0	15.0	14.0
−150	2.0	22.0	22.0	24.0	24.0	27.0	34.0	51.5	55.0	44.0	50.5	50.0	50.0	54.0	57.0	57.0	52.0	45.0

Pebble Load in tube mills is generally kept at slightly above the mill center line, but fluctuations in volume above or below this amount are permissible to allow for increased or decreased tonnage being handled, or to permit variation in grinding fineness. When the level rises considerable above the center line, however, there is a danger of pebbles being "thrown back" from the feed scoop, or of chokage of the discharge screens. Figures showing the difference in horsepower consumption and other data as a result of variation in pebble volume are given in Table LXX. Weight of tube-mill pebble loads is given in Table LXXI.

TABLE LXXI.—WEIGHT OF TUBE-MILL PEBBLE LOADS[65]
In Tons of Pebbles at 105 lb. Per Cubic Foot

Pebble load in 22-ft. tube mill	Internal diameter of tube-mill lining									
	54 in.	55 in.	56 in.	57 in.	58 in.	59 in.	60 in.	61 in.	62 in.	63 in.
12 in. above axis of mill	14.21	14.65	15.10	15.55	16.02	16.48	16.96	17.44	17.92	18.42
11 in. above axis of mill	13.81	14.25	14.69	15.14	15.59	16.05	16.51	16.98	17.46	17.94
10 in. above axis of mill	13.41	13.84	14.27	14.71	15.15	15.60	16.06	16.52	16.99	17.47
9 in. above axis of mill	13.01	13.43	13.85	14.28	14.71	15.16	15.60	16.06	16.52	16.98
8 in. above axis of mill	12.60	13.01	13.42	13.84	14.27	14.70	15.14	15.59	16.04	16.50
7 in. above axis of mill	12.18	12.58	12.99	13.40	13.82	14.24	14.67	15.11	15.56	16.01
6 in. above axis of mill	11.76	12.15	12.55	12.95	13.36	13.78	14.20	14.63	15.07	15.51
5 in. above axis of mill	11.33	11.72	12.11	12.50	12.90	13.31	13.73	14.15	14.58	15.01
4 in. above axis of mill	10.90	11.28	11.66	12.05	12.44	12.84	13.25	13.66	14.08	14.51
3 in. above axis of mill	10.47	10.79	11.21	11.59	11.98	12.37	12.77	13.17	13.59	14.00
2 in. above axis of mill	10.04	10.40	10.76	11.14	11.51	11.90	12.29	12.68	13.09	13.50
1 in. above axis of mill	9.60	9.96	10.31	10.68	11.05	11.42	11.80	12.19	12.59	12.99
Level with axis of mill (*i.e.*, half full)	9.18	9.53	9.88	10.23	10.60	10.96	11.34	11.72	12.11	12.50
1 in. below axis of mill	8.77	9.10	9.45	9.79	10.14	10.51	10.88	11.25	11.63	12.01
2 in. below axis of mill	8.33	8.66	9.00	9.33	9.69	10.03	10.39	10.76	11.13	11.50
3 in. below axis of mill	7.90	8.27	8.55	8.88	9.22	9.56	9.91	10.27	10.63	11.00
4 in. below axis of mill	7.47	7.78	8.10	8.42	8.76	9.09	9.43	9.78	10.14	10.49
5 in. below axis of mill	7.04	7.34	7.65	7.97	8.30	8.62	8.95	9.29	9.64	9.99
6 in. below axis of mill	6.61	6.91	7.21	7.52	7.84	8.15	8.48	8.81	9.15	9.49
7 in. below axis of mill	6.19	6.48	6.77	7.07	7.38	7.69	8.01	8.33	8.66	8.99
8 in. below axis of mill	5.77	6.05	6.34	6.63	6.93	7.23	7.54	7.85	8.18	8.50
9 in. below axis of mill	5.36	5.63	5.91	6.19	6.49	6.77	7.08	7.38	7.70	8.02
10 in. below axis of mill	4.96	5.22	5.49	5.76	6.05	6.33	6.62	6.92	7.23	7.53
11 in. below axis of mill	4.56	4.81	5.07	5.33	5.61	5.88	6.17	6.46	6.76	7.06
12 in. below axis of mill	4.16	4.41	4.66	4.92	5.18	5.45	5.72	6.00	6.30	6.58

The **Speed** of a tube mill is usually calculated from Davidsen's formula, $N = \frac{200}{D}$, where N equals the revolutions per minute, and D the inside diameter of the mill in inches.

Approximate Horsepower consumed in tube milling, where speed is regulated by Davidsen's formula, equals 0.15 M, where M refers to the pebble contents of the mill in cubic feet. Moisture percentage in the feed and other factors will influence the result.

Feed Tonnage in **Tube Mills** usually varies with immediate conditions, and results are interdependent with other factors. A return system for unground product is imperatively necessary where sliming is practised, and the feed should be in such quantity that ample unground material remains near the discharge end to provide work for the falling pebbles, and to prevent undue wear on the liners. Figures showing the effect of tonnage variation in tube-mill feed are given in Table LXXII.

TABLE LXXII.—EFFECT OF FEED TONNAGE VARIATION IN TUBE MILLING[64]

Size of mill		23 ft. × 5 ft.							
Pebble load		20,000 lb.							
Feed		3 in.	3 in.	3½ in.	3½ in.	4 in.	4 in.	4½ in.	4½ in.
Tons ore per 24 hr.		172	172	190	190	216	216	231	231
Percentage ore		64.71	66.67	71.05	67.85	68.18	69.70	66.67	72.22
Percentage solution		35.29	33.33	28.95	32.14	31.82	30.30	33.33	27.78
Indicated h.p.		56.4	54.28	51.60	54.80	53.20	49.40	47.50	43.50
Mesh	Feed	Discharge, Grading Analysis							
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
+ 20	6.0								
+ 30	20.0								
+ 40	24.0								
+ 60	23.0	7.0	13.0	12.5	14.0	16.0	14.0	26.0	30.0
+ 80	11.0								
+100	8.0	32.0	35.0	36.0	34.0	34.0	36.0	38.0	30.0
+120	4.0								
+150	2.0	13.0	11.0	10.0	12.0	14.0	16.0	11.0	10.0
−150	2.0	48.0	41.0	41.5	40.0	36.0	34.0	30.0	30.0

Moisture Percentage in tube-mill feed is generally kept at about 38 per cent., but efficiency figures of actual practice, from which such a figure may be derived, are usually based on operations where tube-mill feed is composed of comparatively fine material. Where very coarse crushing is practised previous to tube milling, it is often found advisable to reduce the moisture below the generally-accepted percentage, as otherwise the lifted pebbles tend to drop the coarser particles of ore attached, to the detriment of crushing efficiency. The general effects of moisture variation are shown in Table LXXIII.

General Tube-Milling Data, referring to the details of operation of these machines in various mills, are given in Tables LXXIV and LXXV.

TABLE LXXIII.—EFFECT OF MOISTURE PERCENTAGE VARIATION IN TUBE MILLING[64]

Size of mill	23 ft. × 5 ft.										
Pebble load	20,000 lb.										
Tons ore per 24 hr.	172										
Ore feed	3 in.										
Ore, per cent.	71.43	69.06	65.67	65.20	63.78	60.70	60.44	55.71	47.10	45.40	38.90
Solution, per cent.	28.57	30.44	34.33	34.80	36.22	39.30	39.56	44.29	52.90	54.60	61.10
Indicated h.p.	45.0	48.9	55.8	57.4	58.0	56.9	55.0	55.8	59.0	62.3	62.3

Mesh	Feed	Discharge, Grading Analysis										
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
+ 20	6.0											
+ 30	20.0											
+ 40	24.0											
+ 60	23.0	12.5	13.0	8.0	8.0	9.0	8.0	7.0	8.0	7.0	5.5	8.0
+ 80	11.0											
+100	8.0	36.0	34.0	30.0	32.0	33.0	33.0	31.0	34.0	32.0	30.5	32.0
+120	4.0											
+150	2.0	10.0	12.0	13.0	14.0	12.0	13.0	13.0	12.0	14.0	13.0	12.0
−150	2.0	41.5	41.0	49.0	46.0	46.0	46.0	50.0	46.0	47.0	41.0	48.0

TABLE LXXIV.—COMPARATIVE

Company	When published	Number of mills	Make	Size	Volume, cu. ft.	R.p.m.	Capacity, tons per 24 hr.
Oroya Brownhill, W. Australia	1904	5	.....	Inside 3′8″×13′7″	143	.....	37.5 thru 200
Hannans Star, W. Australia	1904	..	.....	4′×16′	201	29	38, 95%–150 mesh
Yuanmi, W. Australia	1912	1	.....	4′×16′	201	.....	34, 150
Waihi, New Zealand	1911	3	.....	Inside 4′9″×18′	319	25–26	110, 120
Rand, South Africa	1912	..	.....	5′6″×22′	523	.....	130 thru 90-mesh
Rand, South Africa	—	8	.....	5′6″×22′	523	28	240, 90%–90
Dos Estrellas, El Oro, Mex	1909	4	Allis-Chalmers	5′×24′	471	28	75, 58%–200
Esperanza, El Oro, Mex	1912	10	Krupp	Inside 48¾″×19′8″	250	32	40
Santa Gertrudis, Pachuca, Mex	1912	6	.....	5′×16′	314	.....	.....
		4		5′×20′	392		
Mexico, Mines of, El Oro, Mex	1912	6	Krupp	4′×20′	250	31	50, 60 90%–200
Lucky Tiger, Sonora, Mex	1912	5	P. and M. M. Co.	5′×14′	274	.....	51, 85%–200
Montana-Tonopah, Nev	1908	2	Allis-Chalmers	5′×22′	432	27	52, 66%–200
Goldfield Con., Goldfield, Nev.,	1912	6	A. C.	5′×22′	432	.....	.....
Black Oak, Tuolumne Co., Calif	1912	1	.....	5′×18′	353	23	.....
Liberty Bell, Telluride, Colo	1913	3	Abbé	5′×22′	432	.....	.....
Hollinger, Porcupine, Ont., Can	1912	4	A. C.	5′×20′	392	28	90, 90%–200
Dome, Porcupine, Ontario	1912	4	A. C.	5′×22′	432	32	.....
New Nipissing, Low Grade, Cobalt, Ont	1913	4	.....	6½′×22′	...	.....	.....
Cement Industry	1912	..	.....	5′×22′	432	.....	.....

TUBE-MILL PRACTICE[66]

H.p. consumed	Pebbles	Lb. pebbles, per ton	Cost, pebbles per ton	Kind of lining	Cost of lining	Total cost per ton milled	Remarks
17	.....	.....	.....	Hard iron	.....	$0.421	Power cost $0.099.
.....	5.5 Tons						
30	Flint	.....	.....	.....	.....	0.436	
30–35				Corrugated			Feed 70% + 100 mesh
	Flint	.....	.....	Mn steel	.....	.....	24% − 150
	5.5 Tons			Cast iron			
55	Flint	2.6	$0.017	Ribbed	$0.011	0.094	Feed 10 mesh
75–90	8–14 tons						1½ tons per h.p.
	Quartz	.....	.....	Silex	.....	.....	(from ball) thru 90 mesh
108	13 tons						Feed 3 mesh (0.29″)
	Quartz	.....	.....	Ribbed	.....	.....	Lining lasts 300 days
.....	Quartz	15	.....	El Oro	.....	0.156	
40	Quartz	.....	.....	Ribbed	.....	.....	Feed largely quartz and tailings
.....	.....	.....	.....	.....	.....	.....	.....
.....	Quartz	.....	.....	El Oro	.....	.....	
43–48	Flint	8.7	.....	Ribbed	.....	0.65	Feed 0.75 mm.
42.5		2.2	0.055	4″ Silex	0.069	0.358	Lining lasted 8 mos.
60	.....	1.8	.....	Silex	0.023	0.166	Silex lasts 7 mos.
43	.....	.....	.....	Ribbed	.....	.....	Silent chain drive
.....	Flint	.....	.....	4″ Silex	.....	0.0769	Lining lasts 9–10 mos.
.....	Flint	.....	.....	Silex	.....	.....	Silent chain drive
.....	.....	.....	.....	.....	.....	.....	
.....	.....	.....	.....	.....	.....	.....	
70–80	Flint	.....	.....	.....	.....	.....	Product from ball mills

TABLE LXXV.—TUBE-MILL DATA[49]

	Tons milled per 24 hr.	No. of tubes	Dimensions, ft.	H.p. hr. per ton milled	Cost per ton milled, cents	Final product	
						%—100 (0.0055 in.)	%—200 (0.0029 in.)
Mex. Silver Mill.....	1000	6	5 × 16	....	....	.....	.....
		6	5 × 20	19.2	22.0	98.0	75.0
Nipissing.....	245	4	6 × 20	44.6	50.0	100.0	100.0
Belmont.....	500	7	5 × 18	16.8	40.9	99.0	72.3
Silver Peak.....	500	1	5 × 18	2.4	2.6	64.0	50.0
Homestake.....	4500	2	5 × 14	....	....	.....	.....
		2	5 × 18	0.8	1.2	87.0	66.0
Hollinger.....	585	6	5 × 20	16.0	18.8	100.0	90.0
Porcupine Crown....	150	1	5 × 16	....	....	.....	.....
		1	4 × 20	11.5	36.0	100.0	90.0
Liberty Bell.....	485	3	5 × 22	4.3	7.7	87.0	73.4

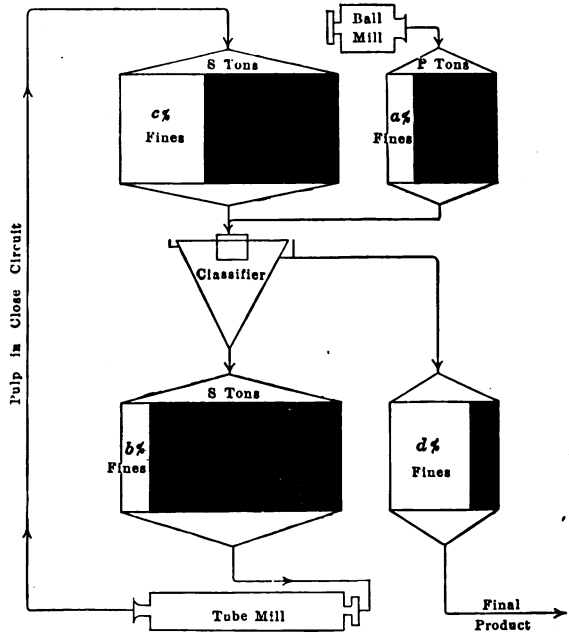


FIG. 129.—Close-circuit diagram.

The **Tonnage** of ore circulating in a **Close-Circuit Grinding System** is a factor in the maintenance of high crushing efficiency. The calculation is, therefore, desirable. A number of formulæ have been evolved by various methods of calculation to aid in estimating the amount by means of screening analyses, as originally suggested by Noel Cunningham. A simple method is to equate tonnage of fine product (*i.e.*, through a certain mesh) obtained by subtracting the amount found in the feed to, from the amount found in the discharge from, the secondary crushers, with the difference in amount in the pulp from the primary crushers and the final product. This is shown graphically in Fig. 129. If P equals tonnage from primary crushers to classifiers and containing a per cent. of material passing a particular mesh; S equals tonnage to and from the secondary plant, entering with b per cent. and leaving with c per cent. of material of the same fineness; and if d per cent. equals the percentage of fines in the final product, then:

$$\text{Tons of fines produced} = Sc - Sb$$
$$= Pd - Pa$$

Therefore S, the tonnage circulating through the secondary plant

$$= \frac{P(d-a)}{c-b}$$

For example: the diagram may be used to represent an instance where the original feed from a ball mill amounted to 40 tons (P) containing 25 per cent. (a), or 10 tons, of material passing a certain mesh. This product joins the discharge from the tube mill, the mixture is classified, and a coarse material containing 12.5 per cent. (b) of fines goes to the tube mill. The percentage of fines increases during grinding to 37.5 per cent. (c), and the final product leaving the classifier is found to contain 75 per cent. (d). Hence: S, the tonnage in close circuit,

$$= \frac{P(d-a)}{c-b}$$
$$= \frac{40(75-25)}{37.5-12.5}$$
$$= 80$$

The circulating tonnage is, therefore, twice the original tonnage —a fact that is visually evident from a glance at the diagram.

SECTION XIII

CONVEYANCE AND ELEVATION OF ORE PULP

Pulp Elevation Problems are governed by the factors of ore coarseness, specific gravity of fluid, and elevation required. Owing to the varied conditions of work and the different characteristics of the material to be elevated, the principal requirement is suitability; *i.e.*, the system and equipment installed should involve minimum attention for control and repair, and must operate at a satisfactory efficiency. Elevation of pulp and water are phases of ore-dressing work in which continuity of operation is of primary importance, and for this reason every opportunity should be taken advantage of in the first instance to install equipment which will do the work efficiently and without unnecessary attention. The question of loss of time to effect repairs and renewals should also be considered. Such apparatus may be divided into nine divisions, according to the class of material to be handled and the height to which it must be elevated. Table LXXVI shows the applicability of eight classes of apparatus for different kinds of work.

TABLE LXXVI.—MILL PUMPING SYSTEMS[67]

Material to be pumped	Low lift (to 25 ft.)	Medium lift (to 50 ft.)	High lift (to 100 ft. or more)
Water or solution	Air lift Frenier pump Centrifugal pump	Centrifugal pump Compound air lift	Plunger pump (1-, 2-, or 3-throw) Compound centrifugal pump
Slime pulp	Air lift Diaphragm pump Frenier pump	Compound air lift Centrifugal pump	Plunger pump (1-, 2-, or 3-throw)
Sand pulp	Centrifugal pump Air lift Frenier pump	Centrifugal pump Tailings wheel	Plunger pump (3-throw)

The simple **Centrifugal Pump** is largely used for the elevation of water and ore pulp. For low lifts it is favored on account of compactness, and because it can be direct connected to an electric motor. Normal velocity for this type of pump is high, even with low lifts. Hence the greater ability of the centrifugal to keep grains of sand in suspension until the discharge point is reached. If there is a tendency for the solid matter in the pulp to settle quickly, the centrifugal should be adopted. It is not suitable if dilution with gland clearance water must be avoided. For medium lifts the centrifugal has its disadvantages. Considerable trouble is often caused by the wear of liners and runners,

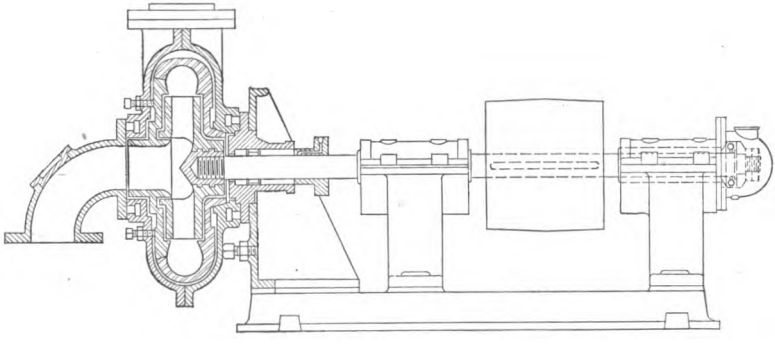


FIG. 130.—Centrifugal sand pump.[68]

on account of the comparatively high speed necessary for effective operation. The wear causes uncertain delivery long before it is time to hang up for repairs. If continuity of operation is desired it is imperatively necessary to erect in duplicate, so that a second pump will always be available in case of breakdown. A standard type of centrifugal pump is shown in Fig. 130.

Centrifugal Pump Runner Speed is designed in definite ratio to height of lift. High lifts involve excessive speed, and this results in considerable friction, and wear on the liners. The single-type pump is, therefore, unsuitable for the purpose. Details of centrifugal pump operation are given in Table LXXVII.

TABLE LXXVII.—CENTRIFUGAL PUMP OPERATION[69]

No. pump (diam. discharge)	Horsepower for each foot head	Nominal capacity gal. per min.	Diameter and face of pulley, inches	Floor space, in.	Shipping weight, pounds	Rev. per minute for given heads						No. pump
						10 ft.	20 ft.	30 ft.	40 ft.	50 ft.	60 ft.	
2	0.1	110	8× 8	46×20	625	505	715	875	1005	1125	1230	2
3	0.22	260	10×10	47×27	1050	400	570	700	805	910	985	3
4	0.35	450	12×10	48×29	1150	435	610	745	865	970	1055	4
5	0.5	700	14×10	59×33	2000	400	570	700	805	910	985	5
6	0.68	1000	14×12	60×38	2400	340	475	580	670	750	815	6
7	0.86	1350	18×14	74×44	3200	320	450	545	635	710	775	7
8	1.2	1800	20×14	75×47	4300	290	410	500	575	640	700	8
10	1.9	2800	22×16	84×50	5200	315	445	540	630	705	770	10
10	1.9	2800	24×16	90×60	7000	240	340	415	480	540	590	10
12	2.5	4000	24×16	92×63	9400	220	315	380	440	490	540	12

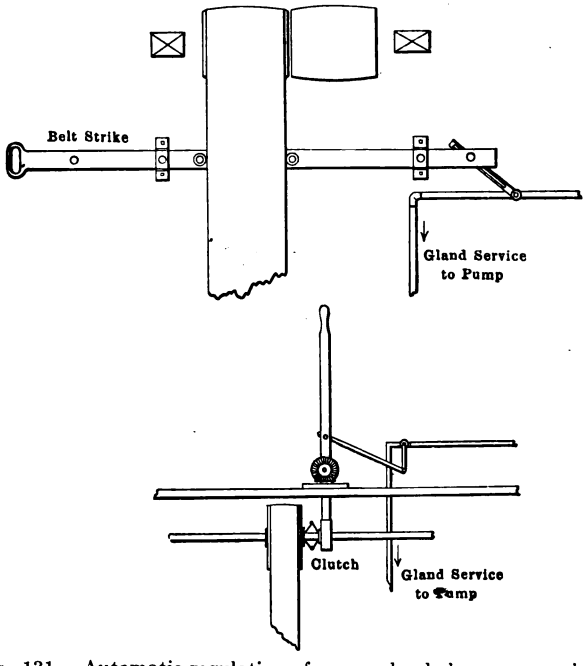


FIG. 131.—Automatic regulation of pump-gland clearance service.[70]

Clearance Water under Pressure is an essential precaution to keep grit from the packing in plunger or centrifugal pumps handling sand or slime. To insure the water being turned on when the pump is started the schemes illustrated in Fig. 131 are recommended.

Diaphragm Suction Pumps (Fig. 132) are largely superseding the air lift for low-lift slime-pulp elevation. An adjustable length of stroke makes regulation satisfactory. The ability

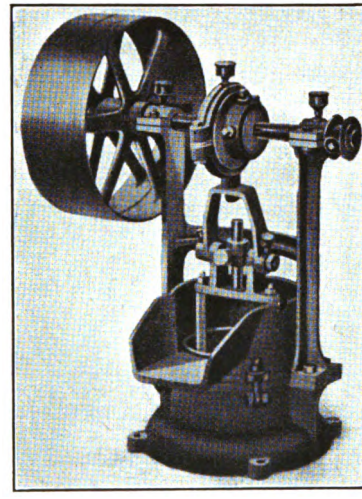


FIG. 132.—Diaphragm suction pump.[31]

of this type of pump to counterbalance the difference between the static head of the average of the thin pulp in a tank or vat, and the thick pulp at the discharge, without loss of fall, has resulted in its wide adoption to handle the discharge from mechanical thickeners, particularly those of the Dorr continuous type. The height of lift is exceedingly limited. Table LXXVIII gives theoretical possible lifts, but owing to the slow speed at which the pump operates and the fact that entangled air is usually present in the pulp there is a difficulty in maintaining the vacuum under normal conditions. The limits in practice

are usually from 50 to 75 per cent. of those given in the table. Heating the pulp or diluent is often a necessary phase of operations in a milling plant. The effect of this on diaphragm-pump operation may be estimated by reference to Table LXXIX.

TABLE LXXVIII.—APPROXIMATE THEORETICAL SUCTION- AND DIAPHRAGM PUMP LIFTS AT NORMAL TEMPERATURES[67]

Altitude, ft.	Specific gravity of fluid							
	1.000	1.100	1.200	1.300	1.400	1.500	1.600	1.700
	Maximum suction lift in feet							
Sea level	34	31	28	26	24	23	21	20
1,000	33	30	27	25	23	22	21	19
2,000	32	29	26	24	22	21	20	18
3,000	31	28	25	23	22	20	19	18
4,000	29	27	24	23	21	20	18	17
5,000	28	26	23	22	20	19	18	17
6,000	27	25	22	21	19	18	17	16
7,000	26	24	21	20	19	17	17	16
8,000	25	23	21	19	18	16	16	15
9,000	24	22	20	18	17	16	16	14
10,000	23	21	19	18	17	16	15	14

TABLE LXXIX.—EFFECT OF TEMPERATURE ON SUCTION- AND DIAPHRAGM-PUMP LIFTS[67]

Deg. Fahr.	Approximate theoretical suction lift in feet, for water, at sea level	Deg. Fahr.	Approximate theoretical suction lift in feet, for water, at sea level
60	34	140	27
80	33	160	23
100	32	180	16
120	30	200	8

Plunger Pumps are compact and suitable machines for ore-pulp elevation. Single-plunger pumps cannot be used, however, with any success with a pulp carrying rapid-settling particles. The three-throw type is to be preferred and is generally used. Hinged clack valves are unsuitable in any case. For continuous opera-

tion a plunger pump may draw from a sump placed below the level of the pump, within suction limits, but a gravity supply into the suction is to be preferred and is almost imperatively necessary with intermittent operation. Pressure clearance water, delivered to the plunger chamber below the packing,

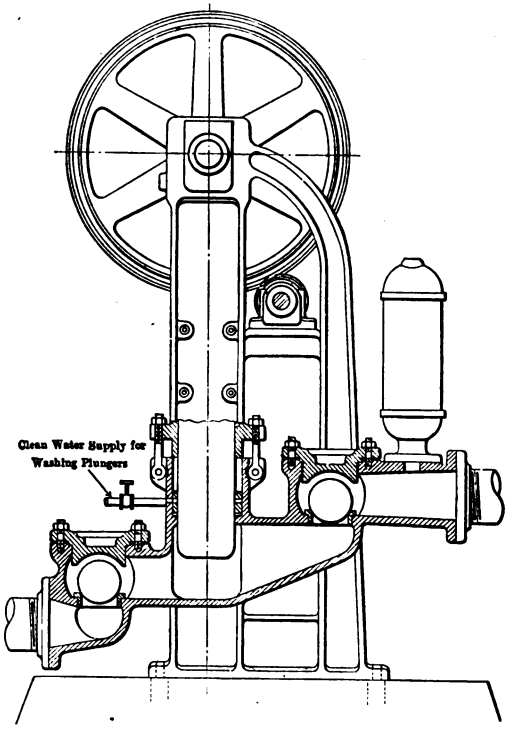


FIG. 133.—Aldrich three-throw slime pump.[71]

is essential to prevent undue wear on the plungers. The speed of the pump is a factor depending upon the character of the solids being lifted. This class of pump is not altogether suitable for coarse-ore pulp, but in the case where it is used for this purpose the velocity of flow should be sufficient to prevent aggregation of sand and consequent chokage in any part of the system.

The **Aldrich Three-Throw Slime Pump** is shown in Fig. 133. Ball valves are used in this pump, and on account of compactness, accessibility of parts, and general construction, the type is to be recommended. Details of operation are given in Table LXXX.

TABLE LXXX.—ALDRICH SLIME PUMP OPERATION[71]

Maximum capacity in U. S. gallons per min.	Maximum pump speed in rev. per min.	Size of pump		Size of suction and discharge	Maximum working lift, 100 ft.	
		Plunger diam., in.	Stroke, in.		Size of tight and loose pulleys, in. Pulleys figured for double belt	Pulley rev. per min. at max. capacity
14	40	3	4	2	10 × 3	200
20	40	3½	4	2½	10 × 3	200
50	40	5	5	3	12 × 3	200
70	40	5	7	4	15 × 3	200
100	40	6	7	5	18 × 4	200
130	40	6	9	5	20 × 4	200
180	40	7	9	6	24 × 5	200
300	40	8½	10	7	30 × 5	200

The Air Lift operates on the principle that a column of ore pulp plus air is lighter than an equivalent column of ore pulp. The lifting action is accelerated by the gradual expansion of the air from maximum to atmospheric pressure, which results in a decrease in the specific gravity of the mixture as it rises in the lift pipe.

A Compound Air Lift consists of two or more simple air lifts arranged in series, the rising main of the first discharging into the sump of the second, and so on. The system can be used to duplicate or triplicate any of the ordinary designs in use. It is suitable where a sump of sufficient depth would be impracticable, or where the available pressure of air is low.

Failure with Air Lifts usually results when the material lifted is too coarse. Efficiency is often sacrificed by using excess air

volume and pressure as a remedy for defective design. Precautions to be avoided have been stated as follows:

(i) An insufficient or an excessive allowance in the ratio of submergence to lift.
(ii) Lack of precautions providing for the intimate mixture of air and pulp at the lift intake.
(iii) The use of bends or elbows in the discharge pipe.
(iv) An unnecessary air pressure or volume.
(v) High velocity in the rising main.

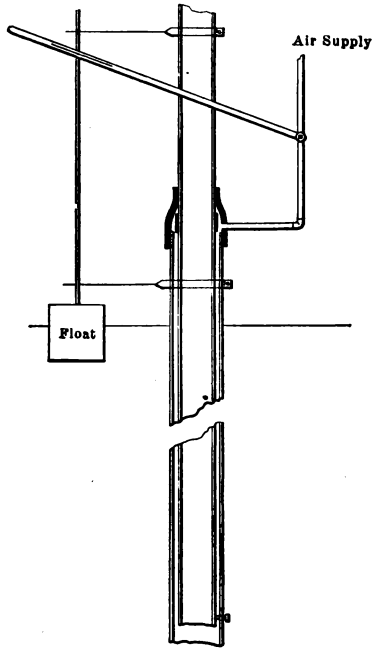


FIG. 134.—Air lift for slime pulp.

A Submergence of from $1\frac{3}{4}$ times to twice the lift has been found to be the most efficient. The lift should be compounded where this submergence is not obtainable. A suitable arrangement for insuring the admixture of air and pulp at the intake is shown in Fig. 134. The air is delivered around the

bottom of the lift pipe and the danger of chokage of the air supply is almost entirely avoided. Where possible the discharge should deliver into the bottom of a short launder having a sharp fall. Pressures required for ordinary air-lift work with various specific gravities of pulp are shown in Table LXXXI. The capacities of air lifts under similar conditions are given in Table LXXXII.

TABLE LXXXI.—THEORETICAL PRESSURES REQUIRED TO ELEVATE WATER AND ORE PULP BY AIR LIFT[73]

Lift, ft.	Submergence, ft.	Theoretical gauge pressure in lb. per sq. in. to elevate fluid of specific gravity					
		Water	Ore pulp				
		1.000	1.100	1.200	1.300	1.400	1.500
5	7.5	3.2	3.5	3.8	4.2	4.5	4.8
10	15.0	6.5	7.1	7.8	8.4	9.1	9.7
15	22.5	9.7	10.7	11.6	12.6	13.6	14.5
20	30.0	13.0	14.3	15.6	16.9	18.2	19.5
25	37.5	16.2	17.8	19.4	21.0	22.7	24.3
30	45.0	19.5	21.4	23.4	25.3	27.3	29.2
35	52.5	22.7	25.0	27.2	29.5	31.8	34.0
40	60.0	26.0	28.6	31.2	33.8	36.4	39.0
45	67.5	29.2	32.1	35.0	38.0	40.9	43.8
50	75.0	32.5	35.7	39.0	42.2	45.5	48.7
5	10.0	4.3	4.7	5.2	5.6	6.0	6.4
10	20.0	8.7	9.6	10.4	11.3	12.2	13.0
15	30.0	13.0	14.3	15.6	16.9	18.2	19.5
20	40.0	17.3	19.0	20.8	22.5	24.2	25.9
25	50.0	21.6	23.8	25.9	28.1	30.2	32.4
30	60.0	26.0	28.6	31.2	33.8	36.4	39.0
35	70.0	30.3	33.3	36.4	39.4	42.4	45.4
40	80.0	34.6	38.0	41.5	45.0	48.4	51.9
45	90.0	39.0	42.9	46.8	50.7	54.6	58.5
50	100.0	43.3	47.6	52.0	56.3	60.6	64.9

In practice an increase over the above pressure figures of from 10 per cent. to 15 per cent. will be required.

TABLE LXXXII.—AVERAGE CAPACITY OF AIR LIFTS HANDLING ORE PULP[73]

Diam. of lift pipe, in.	Cu. ft. per min.	Tons per 24 hr.					
		Water	Pulp of specific gravity				
			1.100	1.200	1.300	1.400	1.500
2	6	270	297	324	351	378	405
2½	10	450	495	540	585	630	675
3	14	630	693	756	819	882	945
3½	20	900	990	1080	1170	1260	1350
4	25	1125	1237	1350	1462	1575	1687
5	40	1800	1980	2160	2340	2520	2700
6	56	2520	2772	3024	3276	3528	3780

The Air Volume required for simple air lifts is shown in Table LXXXIII For a constant length of rising main the amount of free air required decreases as the submergence increases, and increases as the lift increases. The amount of free air required does not vary materially with the weight of the pulp being lifted, other than in the case where excessive velocity in the rising main is necessary to prevent the settlement of solids. In this case the air lift is not suitable. The figures given in the table are approximate. If care is taken in the design of the lift the amount of air actually required will be found to be below the volume given; but on account of the variation of conditions under which such a system may be utilized it is advisable to allow a wide margin.

Altitude does not affect the efficiency of the air lift. The extra volume of free air compressed in the first instance is balanced by the extra work done as a result of the increased expansibility of the air after compression. The air should escape at little above the pressure of the surrounding atmosphere.

The Frenier Pump, or spiral sand pump, as it is sometimes termed, operates on the air-lift principle. The proper operating speed is fixed, and the output from the pump cannot be materially raised or the lift increased by an acceleration in r.p.m. The

TABLE LXXXIII.—AIR VOLUME REQUIRED TO OPERATE AIR LIFT[74]

Lift, ft.	Submergence, ft.	Size of lift pipe, cu. ft. of free air per min.						
		2 in.	2½ in.	3 in.	3½ in.	4 in.	5 in.	6 in.
5	7.5	13	22	31	45	56	90	126
10	15.0	15	24	34	49	61	98	137
15	22.5	16	26	37	53	66	105	147
20	30.0	17	28	39	56	70	112	157
25	37.5	18	30	42	60	75	120	167
30	45.0	19	32	44	63	79	127	177
35	52.5	20	33	47	66	83	133	186
40	60.0	21	35	49	70	86	139	194
45	67.5	22	36	51	73	90	145	202
50	75.0	22	37	52	75	93	150	210
5	10.0	10	17	24	35	43	69	97
10	20.0	11	19	27	39	48	77	107
15	30.0	12	21	30	42	53	85	117
20	40.0	13	23	32	45	57	91	127
25	50.0	14	25	35	49	61	98	136
30	60.0	15	26	37	52	65	104	145
35	70.0	16	28	39	56	69	111	154
40	80.0	17	30	41	59	73	117	164
45	90.0	18	31	43	62	77	123	173
50	100.0	19	32	45	65	81	129	182

air is compressed to a very limited pressure, and satisfactory operation is insured by operating with exactly the amount of pulp required by the pump, allowing free egress from the discharge pipe, limiting the height of lift well within the prescribed figures, and operating on a pulp as fine as possible and carrying no high percentage of heavy mineral. Figure 135 will serve to illustrate the general design, and Table LXXXIV gives average capacities. A deduction, of 4 in. for each 1000 ft., from the stated lift should be made in the case where the pump is to operate at an altitude.

Belt-and-Bucket Elevators are satisfactory machines for the elevation of ore pulp. The design should include provision

for the delivery of pulp into the ascending buckets, an arrangement to provide against slippage due to belt stretching, a safety

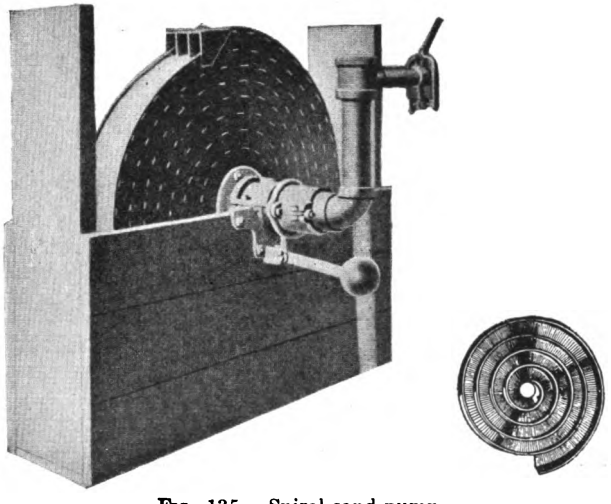


FIG. 135.—Spiral sand pump.

TABLE LXXXIV.—APPROXIMATE FRENIER PUMP CAPACITIES[67]

Size, in.	Specific gravity of fluid					
	1.000	1.100	1.200	1.300	1.400	1.500
	Tons per 24 hr.					
44 × 6	450	490	540	580	630	670
48 × 6	450	490	540	580	630	670
54 × 6	450	490	540	580	630	670
44 × 8	540	590	650	700	760	810
48 × 8	540	590	650	700	760	810
54 × 8	540	590	650	700	760	810
44 × 10	630	690	760	820	880	940
48 × 10	630	690	760	820	880	940
54 × 10	630	690	760	820	880	940

These capacities are based on moderate lifts, precise adjustment and maintenance of level of liquid in the box, and free discharge. If any one of these conditions is impracticable the figures should be reduced accordingly.

boot overflow discharge below the level of the lower bearings, an ample allowance in height for top discharge, and liner plates where wear and scouring action is greatest.

The Buckets on Belt Elevators are sometimes placed in a staggered position. This results in a more even load on the belt and less strain on the center line.

Tailing Wheels are costly to erect and clumsy in appearance, but cheap to operate. They are seldom adopted in countries where mills are covered in. The efficiency of a tailing wheel var from 50 to 60 per cent., and is uniform. Under many conditions it can successfully compete with the centrifugal in the elevation of mill pulp.

Data Dealing with Inclination and Capacity of Launders and piping will be found in Tables LXXXV, LXXXVI, and LXXXVII.

TABLE LXXXV.—LAUNDER GRADES AT GOLDFIELD CONSOLIDATED[76]

Product	Dilution	Width of launder, in.	Height of launder, in.	Grade, in. per ft.	Dry tons per 24 hr.
4 mesh from batteries	4:1	8	8	1¾	255
12 mesh from batteries	6.5:1	8	8	⅝	200
−4 mesh from Chileans	3.3:1	8	8	1⅛	400
−30 mesh from Chileans	3.3:1	8	8	⅝	400
Tube-mill discharge 50 per cent. −200 mesh	1.5:1	6	5	1¾	100
Mixture of tube-mill and Chilean products	2.6:1	8	8	¾	700
Mixture of tube-mill and Chilean products	2.1:1	6	5	1⅛	200
Final product from tube mills	3:1	5½	8	⅜	330
Product from Callow tanks	3:1	3½	3½	⅞	22
Concentrate	9:1	3	3	1 1/16	35
Middling	3:1	3	3	¾	10
Tailing	5:1	5	5	¾	52½
Concentrate	9:1	4	10	¾	50
Tailing	5:1	10½	10½	3/16	800
Water	.....	18	14	⅛	.....

TABLE LXXXVI.—LAUNDER DATA: CANANEA CONSOLIDATED COPPER COMPANY[75]

Material handled	Size of launder, inches, depth × width	Grade, in. per ft.	Lining	Ratio solids: liquids	Largest particle, mm.	Per cent. wt. on 60 mesh
Original feed, to Section "C"	10 × 12	3½	C.I.—11¼"	1:1	30.0	82.0
Bull jig tails, to coarse rolls	10 × 9½	3	C.I.— 9¼"	1: 1.1	28.0	98.0
Roll jig tails, to fine rolls	10 × 7½	2¼	C.I.— 7¼"	1:3	10.0	98.0
Coarse jig concentrates	10 × 7½	1⅜	C.I.— 7¼"	1: 2.9	35.0	95.0*
Undersize 2 mm. trommel to classifier	10 × 5¾	1½	C.I.— 5¼"	1: 4.4	4.0	52.0*
First spigot of classifier to sand jig	8 × 5¾	3⅛	C.I.— 5¼"	1: 1.5	4.0	90.0
Shaking launder[1] for sand jig concentrates	6½ × 12	⅜	None—corners	1:19.0	4.0	84.0
Fine jig concentrates from shaking launder	10 × 7½	1 13/16	C.I.— 7¼"	1: 6.0	4.0	80.0
Bryan (Chile) mill discharge to drag belt	8¾ × 11½	9/16	None—corners	1: 1.7	2.5	26.0
Drag belt sands to classifier distributor	10 × 7½	1⅜	C.I.— 7¼"	1: 0.9	2.5	46.0
No. 1 spigot of classifier to mud jigs	8 × 5½	1⅜	C.I.— 5¼"	1:17.0	3.0	60.0
Table feed, tables 27 to 30	7¼ × 5½	½	None—corners	1: 8.4	1.0	0.3
Table concentrates, drag belt launder[2]	9½ × 9½	⅛	1¼" boards	1:26.0	1.0	13.6
Table concentrates, shaking launder[3]	4 × 7½	3/32	None—corners	1: 7.4	0.5	35.2
Slimes, Section "C," settling tanks to Section "B" vanners	6½ × 9½	9/16	None	1:96.0	0.17	0.0
Slime feed to vanners, Section "C"	9½ × 11½[4]	⅛	None	1: 4.7	0.17	0.5
Vanner concentrates, drag belt launder[5]	10 × 25 × 16	Level	None	1:31.0	0.2	1.0
Vanner and table concentrates	6½ × 6	1 15/16	None	1:20.0	2.0	27.2
Vanner tails, section "C"	9½ × 11½	⅛	None—corners	1: 7.0	0.4	0.4
Coarse tails, table and jig	9½ × 22	⅛	½" boards & corners[6]	1:27.0	2.0	41.6*
Coarse sand tailings to dam No. 2	12¼ × 11½	1 1/32	½" boards & corners[6]	1:21.0	2.0	19.5*
Coarse sand tailings to dam No. 1	12 × 11	9/32	½" boards & corners[6]	1:21.0	2.0	19.6*
Slimes to mill No. 4	9½ × 12½	¼	None—corners	1: 4.4	0.1	0.0
Sands and slimes to mill No. 3	10 × 11½	¼	None—corners	1: 5.0	1.5	2.6
Feed to vanners, mill No. 4	6 × 7½	⅛	None	1: 4.4	0.1	0.0
Slime concentrates, mill No. 4, drag belt launder	12 × 24 × 18	Level	None[2]	.....	0.1	0.2
Slime concentrates, mill No. 4, elevator to bins	7 × 7½	3/16	Glass—corners[7]	1:20.0	0.1	0.4
Slime concentrates, mill No 3, elevator to bins	10 × 12	1 5/16	Concrete	1: 8.7	0.2	2.8
Slime tails, mill No. 4, to mill No. 3 settling tanks	10 × 11	7/32	None—corners	1: 6.3	0.1	0.0

All cast-iron liners have 2-in effective depth and 24-in. length, with corners rounded.

* Liable to choke. [1] Speed 160 r.p.m., actuated by heavy head motion. "Corners" are strips of wood with cross-section of 45° triangle nailed in corners of launder. [2] Speed 75 ft. per min.; old 4-in. drive belts are used, no scrapers. [3] Frequency 180 s.p.m. [4] Effective width, 6 in. [5] Speed 75 ft. per min. [6] Boards laid with grain across direction of flow. [7] Glass liners, ¾ × 4 × 14 in

TABLE LXXXVII.—PULP TRANSFERENCE IN PIPING AT THE HOMESTAKE MILL[81]

	Sand line at Lead	Central City tailing line	Slime line at Lead
Diameter of pipe, in.	12	8	12
Thickness of pipe, in.	9/16	9/16	9/16
Average grade, per cent.	2½ to 5	1¾	1½
Bends in line.	The pipes generally follow the contour of the hills: the sharpest curve is 22½ deg., a standard elbow for flanged C. I. piping being used		
Fineness of tailing in pulp, per cent.	30+100 25+200 45−200	25+100 20+200 55−200	100–200
Water in pulp, per cent.	70−75	85−90	65–70
Solids per 24 hr., tons.	1950	450	1100
Pipe runs.	About ⅓ full	Full	
Wear.	It is customary to turn pipe through about 120 deg. when bottom is worn thin; thus the pipe will be used until three lines of wear have been developed.		
	On 5 per cent. grade worn out in 2 years. On 2½ per cent. grade worn from 9–10 years. Most of line still in use after 11 years, but badly worn.*	In use 8 years without turning	In use 5 years without turning

* Remarks: The grade has been reduced to 2½ per cent. throughout and new piping 1 in. thick has been installed.

SECTION XIV

CLASSIFICATION OF ORE PULP

Classification of **Ore Pulp** is a process relying on specific gravity differences in the ground material, the settlement resistance factor between products having different physical characteristics, and the fluidity of the vehicle in which the ore is temporarily suspended. Colloids in suspension generally interfere with classification results by increasing the viscosity of the liquid.

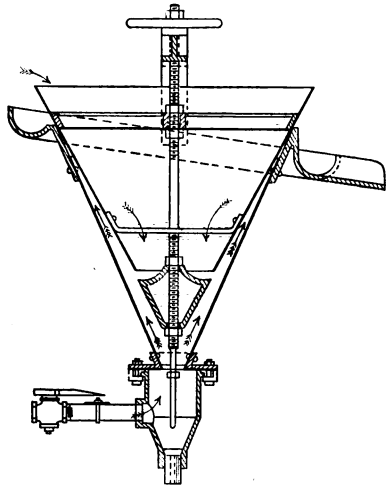


FIG. 136.—Hydraulic classifier.

Classifiers may be divided into two kinds, viz.: the gravitational and the mechanical. In the former class there are numerous designs of the conical or pyramidical form, some of which make use of hydraulic water for the better separation of material of different grades (see Fig. 136) and all relying on the flow of

water carrying the ore to retain the finer particles in suspension. Cone classifiers may be operated "dry" (*i.e.*, with low moisture percentage in underflow), as in the case where a deposit is allowed to accumulate and remain in the cone before any underflow is tapped; or they may be operated "wet" with varying percentages of water in the underflow. Cones running "dry" are usually fitted with a baffle plate near the apex. The function

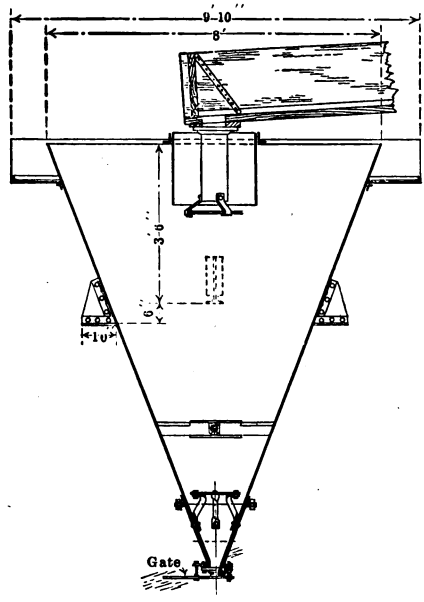


FIG. 137.—Caldecott cone.

of this is to prevent "running" of the entire contents of the cone by "piping" through the center. The Caldecott cone (Fig. 137) is of this type.

The **Allen Cone** is an apparatus of the Caldecott type which is provided with automatic control of underflow. The pulp stream enters the feed spout *A* (see Fig. 138) and, after passing through the truncated cone *B*, the bulk of the water and finer material flows upward and into the overflow launder. The sand

particles settle in the cone and form the basin. When the settled sand reaches the outlet of *B* it obstructs it, so that the water rises in *B*. This lifts the float *C*, thereby removing the ball *G* from the spigot by means of the lever *D*, the link *E*, and

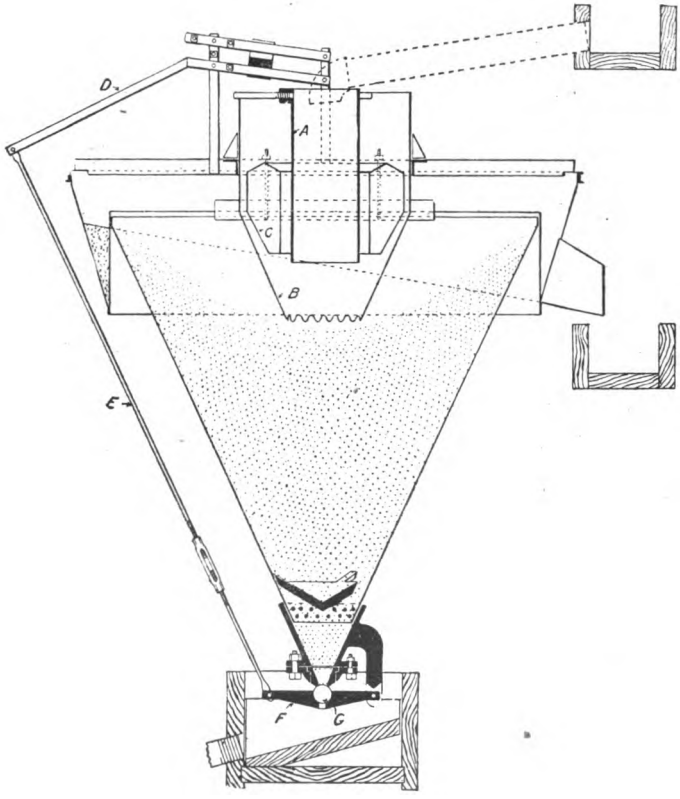


FIG. 138.—The Allen cone.[77]

the valve arm *F*. The settled solids flow out in a continuous stream, the quantity discharged being controlled by the quantity settling in the cone. When sand ceases to enter the cone *B* the ball *G* closes the spigot, and it remains closed until the feed of sand is renewed. The automatic operation of this apparatus

should make it of especial value in those cases where dewatering or classifying by cone has been proved to be cheap and comparatively efficient.

Cone Classifiers of the Multiple Type (Fig. 139) produce more than two products, and similar results may be obtained by using single cones in series. Dimensions of cone parts, and the degree of inclination of the sides are matters depending on character of feed, character of product required, and tonnage to be handled, and are influenced by the physical properties of the ground material. Classification in cones is being carried out under an almost infinite variety of conditions as to design, and no general rules can be enunciated.

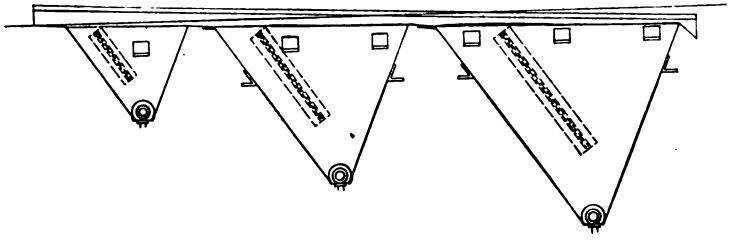


FIG. 139.—Multiple cone classifiers.

Homestake Cone Practice (Fig. 140) represents an unusual arrangement notable for the results obtained and the small amount of attention required. The first section of the installation consists of 14 gravity cones, 4 ft. in diam., with a 70-deg. slope. The usual overflow arrangements are provided, but the underflows are each supplied with a high-pressure emergency water supply, as illustrated, and are each connected by a length of steam hose to a second cone of 16-in. diam. and 80-deg. slope. The latter series of cones are supplied with extra water in the manner shown, and the sand feed is delivered in an upward direction from the lower part of the cone near the apex. The overflow arrangements are as usual, and the underflow passes direct to the tube mill, being diluted with low-pressure water as found necessary.

Spitzlutten are pointed boxes with an ascending current of water to assist classification. An apex discharge is arranged for the coarse product and an overflow, either around the

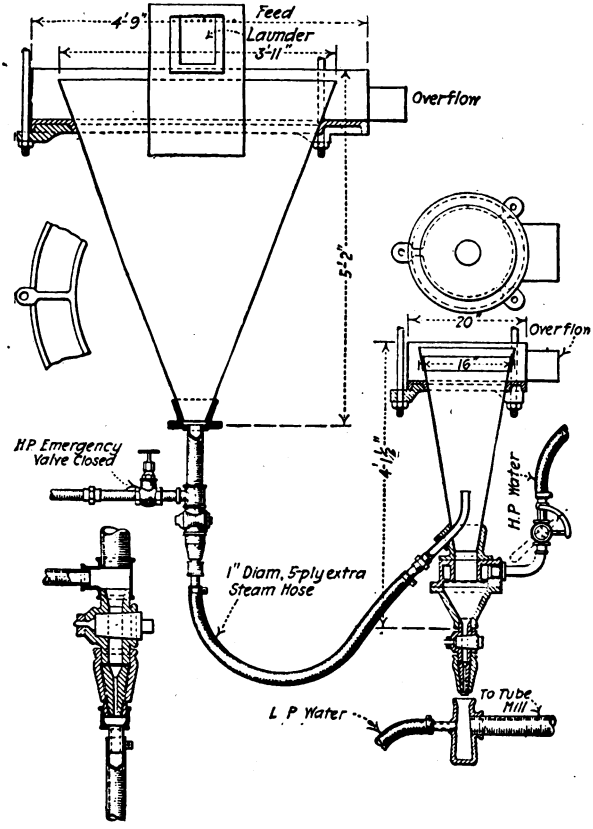


FIG. 140.—Homestake cone practice.[51]

whole or part of the top periphery, carries with it the finer material.

Spitzkasten are similar to spitzlutten but without the ascending current of water. Types are illustrated in Fig. 141.

Porcelain or Glass Nozzles have been used in some mills in the regulation of cone classifier discharge underflow and have been found to outlast steel bushings.

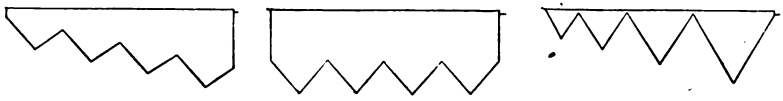


FIG. 141.—Types of compound spitzkasten.

In the **Flood Classifier** (Fig. 142) the upward current of water is applied through the medium of a centrally located pipe and is distributed from three tiers. Coarse material is subjected

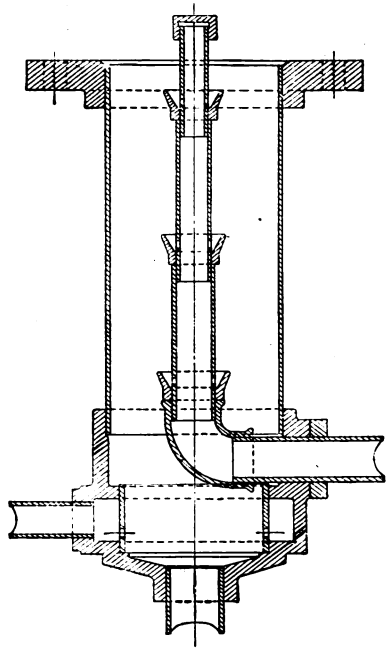


FIG. 142.—Flood classifier.[19]

to the influence of the currents produced during its passage from intake to exit, and fines are separated from coarse material during the process. The classifier may be attached to the bottom

of a launder carrying pulp, or may be fitted with a box top. It may be used singly or in series.

The **Deister Classifier** (Fig. 143) consists of a cast-iron barrel carrying inverted truncated cones arranged with slots of a size permitting the free passage of the coarser material. The pulp

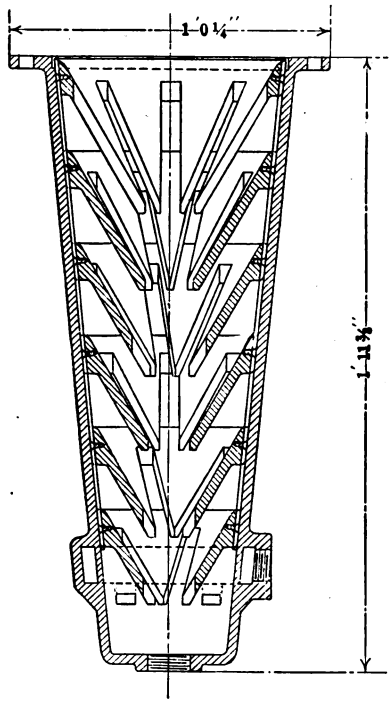


FIG. 143.—Deister cone classifier.[78]

route through these cone slots is staggered, the openings from one cone being out of line, vertically considered, with the adjacent cones. Hydraulic water is admitted through one of the side inlets in the annular waterway underneath the bottom. These cones may be used singly or in series, and their introduction has marked a decided advance in the practice of cone classification.

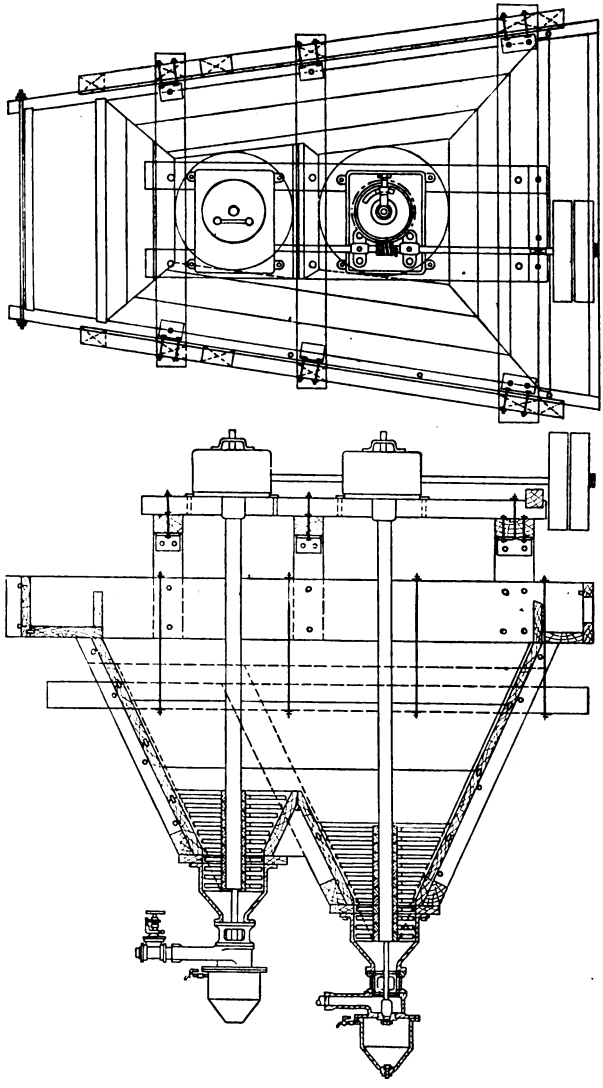


FIG. 144.—Richards-Janney classifier (2 compartment).[11]

The **Richards-Janney Classifier** consists of a series of rectangular compartments in the form illustrated in Fig. 144. In the cone apex a cylindrical settling chamber is arranged which is connected, lower down, with a hydraulic water supply.

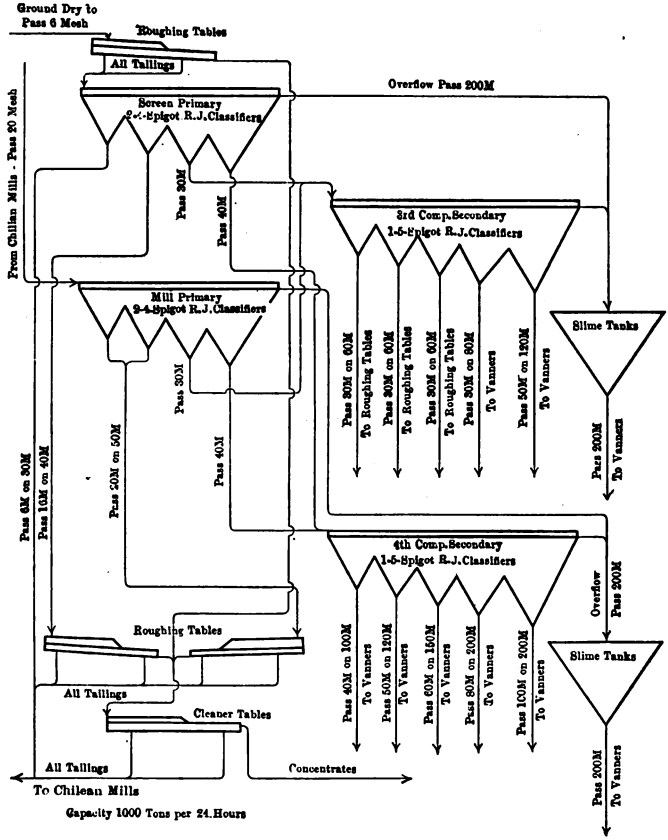


FIG. 145.—Richards-Janney classification practice at the Magna plant of the Utah Copper Company.

A worm shaft, attached to the upper framing, operates at from 80 to 90 r.p.m.; and, by the movement of a shaft connected to a series of sifting blades at the bottom of the chamber, the pulp

is loosened. A valve rod which passes through the spindle shaft, which is hollow, is raised intermittently by cam action, and an amount of material is thus permitted to pass to the hydraulic chamber. On the dropping of the valve rod the classified material passes out and the chamber is then ready for an additional charge.

Richards-Janney Classification at the Magna plant of the Utah Copper Company, Garfield, Utah, is illustrated by flow-sheet (Fig. 145) showing arrangement of classifiers in the mill, accompanied by a statement of comparative results in Table LXXXVIII.

TABLE LXXXVIII.—COMPARATIVE RESULTS OF OPERATION OF RICHARDS-JANNEY CLASSIFIERS

Classifier	Tons handled	Max. mesh	Size bushings, in.	Gal. water entering per ton	Gal. water injected per ton	Gal. water discharged per ton	Gal. water overflow per ton	Total gal. per ton
Screen primary.....	450	6	1½, 1½ 1, 1	225	364	361	228	589
Mill primary.....	757	20	1¼, 1 1, 1¼	361	160	189	314	503
Third compartment secondary.....	233	30	¾, ⅝, ⅝, ⅞, ⅞	824	470	359	964	1323
Fourth compartment secondary.....	310	40	⅞, ⅞, ⅞, ⅞, 1	506	486	463	529	992

NOTE.—The small amount of water injected per ton and discharged per ton in results obtained from mill primary classifier is due to heavy circulation from first and second spigots. See flow sheet (Fig. 145), upon which above data are based.

The launder-type **Richards Pulsator Classifier** consists of a settling pocket which may be attached directly to the bottom of a launder in the manner shown in Fig. 146. Below the settling pocket is a sorting column that receives water by means of a pulsator valve in an upward direction. A spigot discharge is arranged.

In the inverted type **Richards Pulsator Classifier** shown in Fig. 147 the finest material is delivered from the first compartment. Hydraulic water is pulsated into the machine, passes through a separating screen, and through the outlets which are placed only slightly below the level of the feed intake. The various compartments, from the first onward, decrease in cross-section, and as the hydraulic water is distributed under even pressure, the velocity of ascending water in any compartment will vary inversely as its cross-section, being at a minimum in the first compartment and at a maximum in the last compartment. The screen serves to support the ore bed and distribute the water supply evenly. Hydraulic water supply may be adjusted so that all solid particles pass through the discharge spouts; or the spigot just above the screen in the last compartment may be used as an outlet for concentrate.

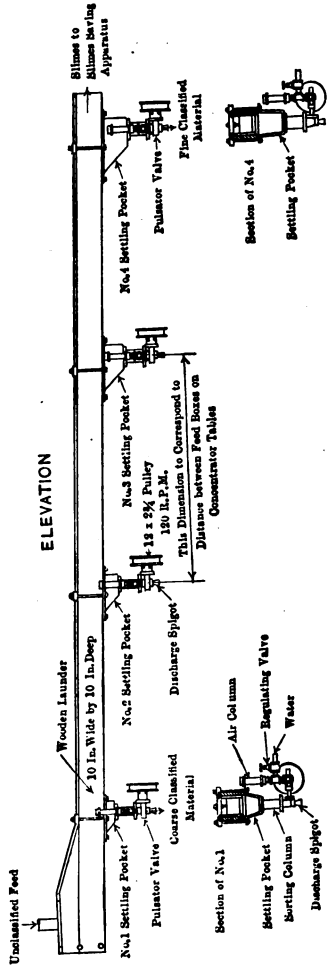


FIG. 146.—Launder-type Richards pulsator classifier.[16]

The **Overstrom Classifier** (Fig. 148) operates somewhat similarly to the Richards Pulsator Classifier and discharges the products at the top of the machine. Referring to the accompanying cut it will be seen that the feed enters at *A*, passes a screen and enters the first compartment, which is sup-

plied with hydraulic water as shown. The slime overflows, and the sand then drops through a slot into the first sorting column. The heavier material drops through *G* and passes to the next compartment, and so on.

In the **Richards Hindered-Settling Classifier** (Figs. 149 and 150) the sorting is effected by permitting the articles to aggre-

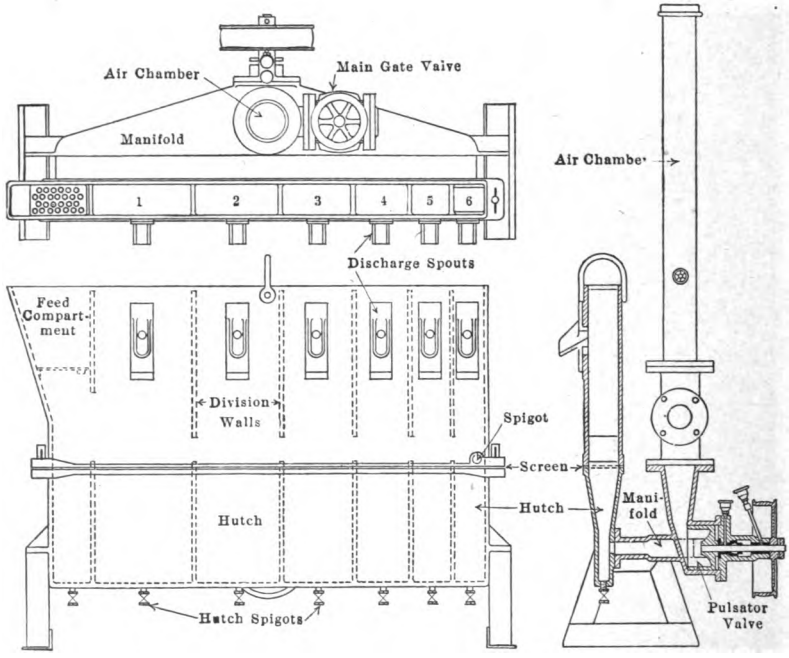


FIG. 147.—Richards pulsator classifier (inverted type).[16]

gate in a constriction formed in the lower part of the sorting column, where they are subjected to the separating effect of an upward current of water.

Concentrating Tables may in certain instances be used as classifiers. The Wilfley table has been extensively used in this respect without any interference with its concentrating functions. A clear line of demarcation between sand and slime zones at the

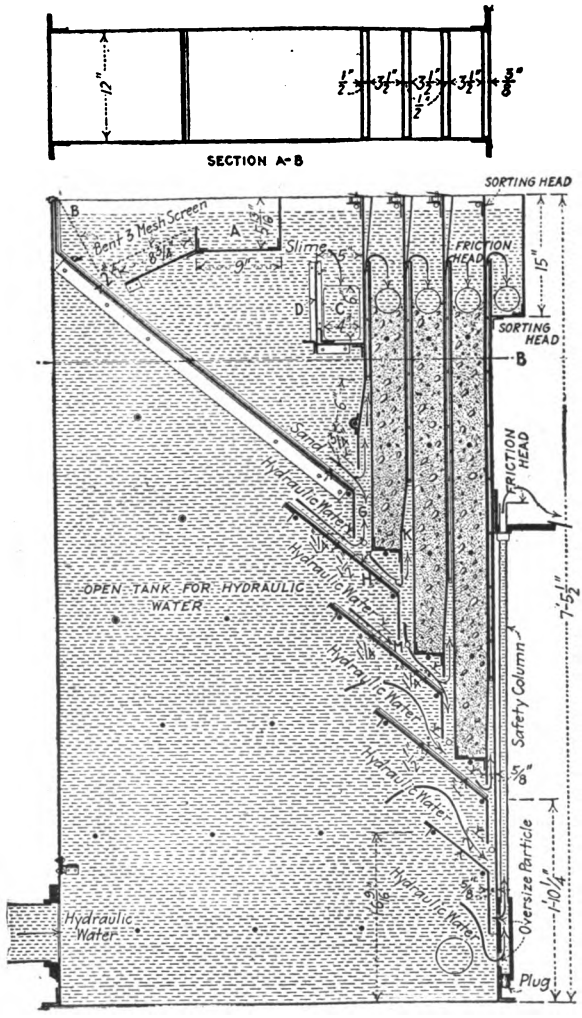


FIG. 148.—Section of Overstrom classifier.[79]

table discharge is generally observable, and advantage is taken of this by dividing the tailing launder and equipping this with an adjustable divider to allow for tonnage fluctuations or variations in the character of the ore.

The **Dorr Classifier** (Fig. 151A) consists of a longitudinal trough with inclined bottom and vertical sides, in which numbers of mechanically operated scrapers serve to drag the coarser material toward the upper end of the trough and expose it to a washing action in so doing. The action of the scrapers is inter-

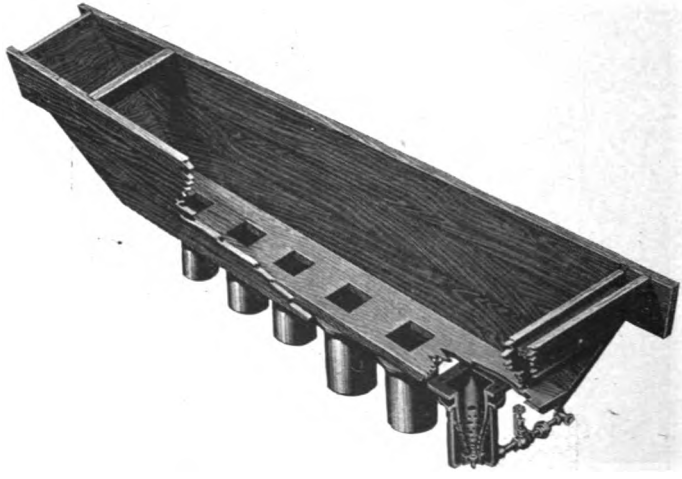


FIG. 149.—The Richards hindered-settling classifier.[11]

mittent and the operation is governed by an arrangement of cams, levers, rods, and cranks, a combination that produces a slow raking motion. A modification of the standard machine—the bowl-type classifier—has a classifying chamber (see Fig. 151B) connected to the classifier tank compartment. The feed is delivered to the center of the bowl, the bottom of which is swept by a mechanism similar to that used in the Dorr thickener. The settled solids are delivered through an orifice to the classifier proper, where the sand is handled in the usual way. Water is

added in the tank compartment. This insures a rising current through the orifice and assists efficient separation.

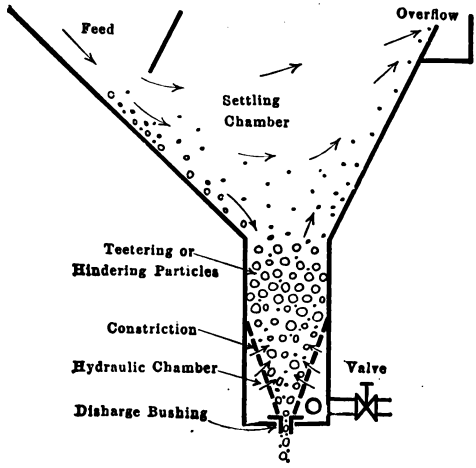


FIG. 150.—Operation of Richards hindered-settling classifier.[11]

The **Akins Classifier** (Fig. 152) operates on the principle of the ribbon conveyor with a large-diameter spiral that prevents

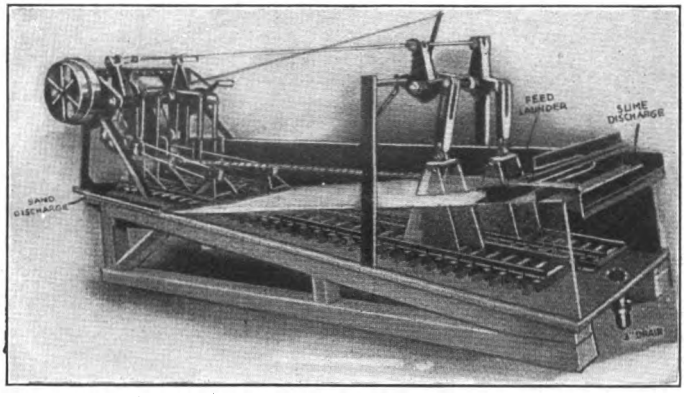


FIG. 151A.—Dorr classifier.[80]

the building up of solids in any part of the apparatus. The helix is continuous for a part of the length commencing from the over-

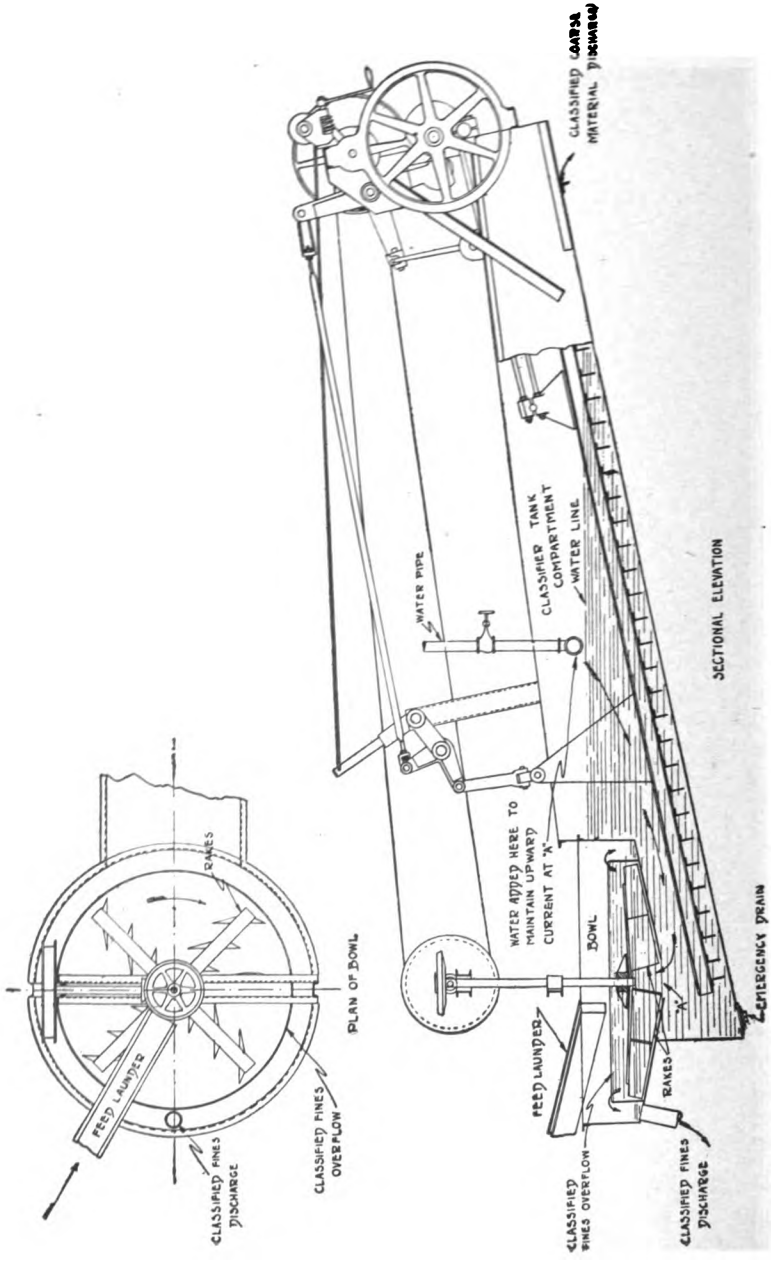


FIG. 151B.—Dorr bowl-type classifier.[80]

flow end, then double and interrupted. This aids in the elimination of moisture in the sand product and assists, by continually turning over the material, in the separation of the required products. The operation of this type of classifier in closed circuit

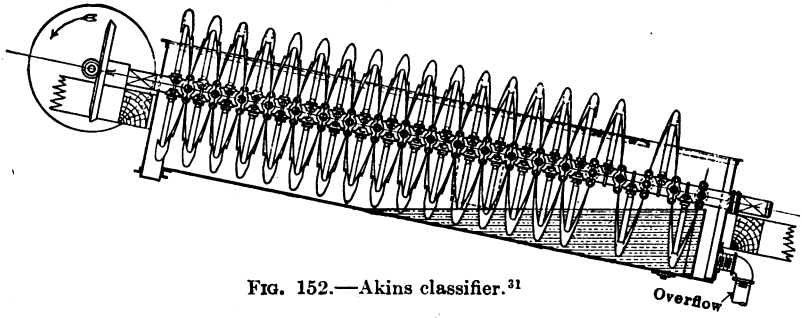


FIG. 152.—Akins classifier.[31]

with a ball mill, and in connection with which no elevating return system is necessary, is shown in Fig. 153.

The **Ovoca Classifier** (Fig. 154) works on the principle of the screw conveyor. The smaller types are fitted with one screw only, whereas the larger ones operate with two screws. The set-

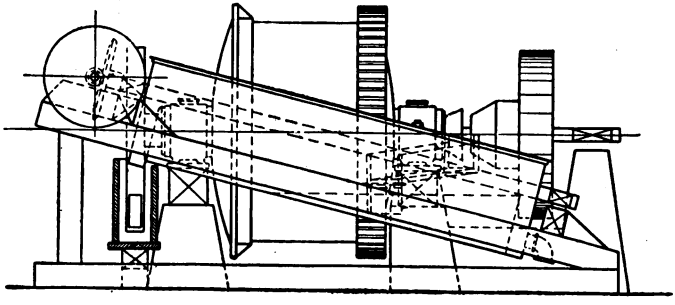


FIG. 153.—Akins classifier in closed circuit with ball mill.[31]

tled ore is withdrawn from a small tank by means of the conveyor and delivered above the classifier intake level. The finer material overflows around an annular discharge attached to the tank.

The **Federal Esperanza Classifier** consists of a series of scrapers attached to a belt which remove and elevate the settled material from the floor of a trough with inclined bottom.

The **Drag Classifier** used at the **Inspiration** mill handles 800 tons per day and consists of two 18-in. belts with scrapers attached. The returning belt is raised to a point above the level of the pulp by means of a duplication of the pulley at the feed end, a second pulley being placed vertically above the other.

Classifier Efficiency is seldom determined, owing to the prevalent opinion that the work of the machine may be gauged from a statement of grading analyses of feed and discharge. As a matter of fact such figures are always valueless and sometimes mis-

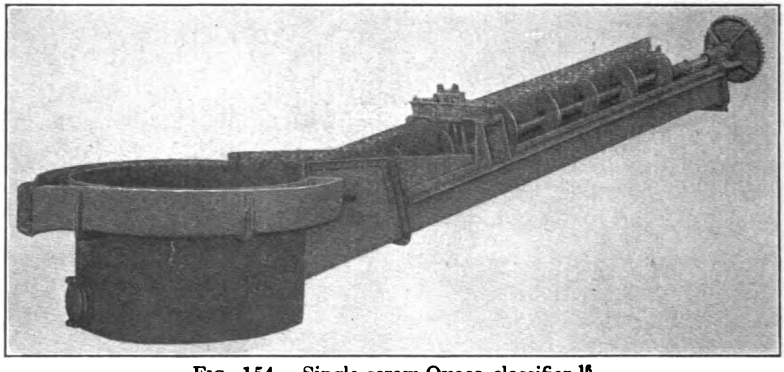


FIG. 154.—Single-screw Ovoca classifier.[16]

leading except when used in conjunction with tonnage of feed and individual discharge. With simple classifiers the efficiency of classification at a certain mesh, dividing an undersize from an oversize, may be found from the following formula:

$$\frac{100(t + t')}{F} = \text{efficiency of classification,}$$

where t = tonnage of oversize in the coarse product, and t' = tonnage of undersize in the fine product, and F = total feed tonnage, all per unit of time. An example is given in Table LXXXIX.

Compound-Classifier Efficiency may be estimated in a similar way by extending the formula thus:

$$\frac{100(t + t' + t'' \ldots\ldots + 0)}{F} = \text{efficiency of classification,}$$

where t = tonnage of required grading in first product, t'' = tonnage of required grading in second product, etc., and 0 = tonnage of required undersize in overflow, and F = total tonnage in feed.

TABLE LXXXIX.—CLASSIFIER EFFICIENCY

Plant: Inspiration Consolidated Copper Co.
Classifier: Model D Dorr, 6 × 27 ft.
Speed: 25 strokes per min.
Slope: 2⅝ in. to 1 ft.
Tonnage feed: 1100
Tonnage coarse product discharge: 790
Tonnage fine product discharge: 310
Separation desired: 65 mesh

Screening analysis

Mesh	Coarse-product discharge, per cent.		Fine-product discharge, per cent.	
+ 28	32.2			
− 28 + 48	39.6		0.6	
− 48 + 65	13.1		5.2	
		84.9		5.8
− 65+100	6.6			
−100+150	2.9			
−150+200	1.3			
−200	4.3			
		15.1		94.2
		100.0		100.0

Classification efficiency: 84.9 per cent. of 790 tons = t
94.2 per cent. of 310 tons = t'
= 962.73 tons of a possible 1100 = 87.5 per cent. efficiency.

The **Elimination** of **Oversize** from **Overflow** is sometimes of prime importance. In other cases the inclusion of fines in the underflow is to be avoided. In either case the simple method of calculating classifier efficiency outlined will, if utilized, serve to assist adjustment and produce the most satisfactory results possible.

SECTION XV

THICKENING, SETTLING, AND DEWATERING OF ORE PULP

Settlement and **Thickening** of **Ore Pulp** are generally synonymous phrases. Apparatus used may be considered under two headings, viz.: the gravitational, and the gravitational-plus-mechanical. Among gravitational settlers may be mentioned the conical and spitzkasten types, or simple sloping-bottomed vats. The gravitational-plus-mechanical types have an arrangement whereby discharge is assisted and regulated by mechanical means, with either an intermittent or continuous action.

Gravitational-Thickening Efficiency is governed by various factors, among which may be mentioned, (*a*) dilution of pulp feed, (*b*) condition of colloidal constituents with reference to electrolytic coagulation, (*c*) area of settlers, and (*d*) depth of settlers. Assuming maximum favorable comparative conditions in these respects, the actual degree of thickening attained with a fine product will be proportional to the time allowed. Where maximum dewatering is required the intermittent gravitational-plus-mechanical type of settler will give a product freer from water than the continuous machine. The advantages of ample allowance in the time factor for slime settlement may be utilized without direct mechanical aid, as in the case where the settled contents of large sloping-bottomed vats are sluiced out with the cyanide solution needed for subsequent recovery treatment. Intermittent gravitational thickening is only feasible in large-capacity vats.

The **Pulp Thickener** illustrated in Fig. 155 is of the intermittent type and is operated in series. As soon as filled with slime, *i.e.*, when the overflow is no longer clear, the pulp stream is

diverted. The pulp in the vat is allowed to thicken to any desired extent and the clear water decanted off. An agitator paddle, at the end of a square vertical shaft and driven by a crown-wheel gearing through which the shaft is free to move in a vertical direction, is then slowly lowered into the pulp,

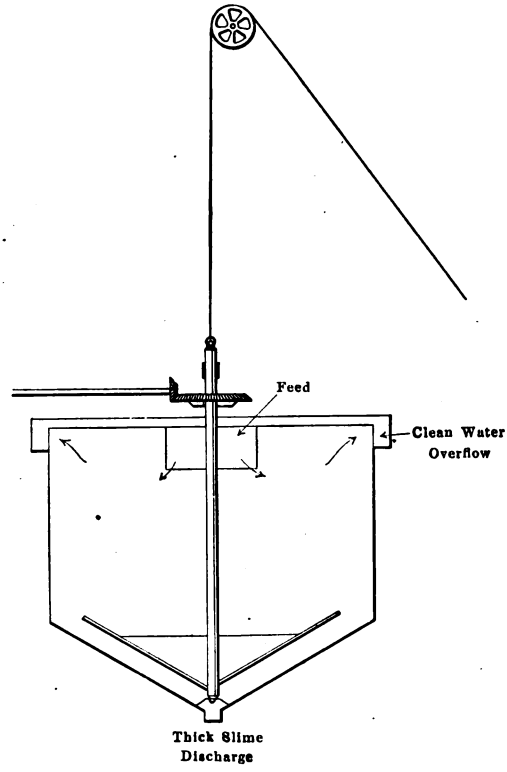


FIG. 155.—Pulp thickener.

revolving the while at about 20 r.p.m., until a few inches from the bottom. After a few moments the contents of the vat are thoroughly mixed and the thick sludge may be drawn off through an appropriate discharge valve attached to the cone at the bottom of the vat.

The **Dorr Continuous Thickener** (Fig. 156) consists of four radial arms attached to a central vertical shaft, suspended in a flat-bottomed tank in the manner shown, and revolved by means of worm gearing at a slow speed. The ploughs scrape the settled slime toward the center of the tank, where it is discharged or conveyed elsewhere. The discharge is preferably handled by means of a diaphragm pump, as shown in the

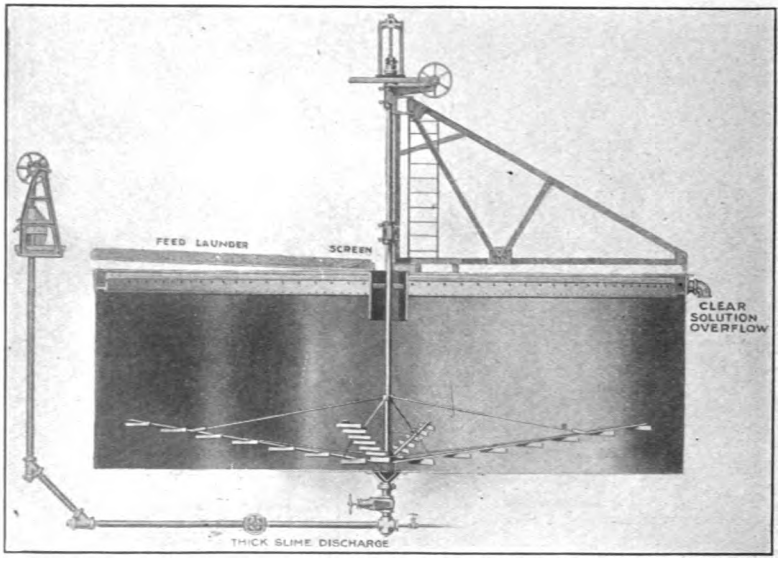


FIG. 156.—Dorr continuous thickener.[80]

cut, whereby loss of fall is avoided and improved results are insured. Details of typical practice are given in Table XC.

The **Dorr Tray Thickener** is an adaptation of the standard machine whereby increased settling area is obtained by means of a division or divisions of the tank into horizontal compartments, each fitted with a bottom tray conforming to the slope of the additional plough mechanism, which is a duplicate of that found in the ordinary machine. Two types are now in operation.

The **Open-Type** thickener is shown in Fig. 157. The **Connected-Type** thickener is arranged for the separate delivery of feed to each section, and the individual discharge of thickened products.

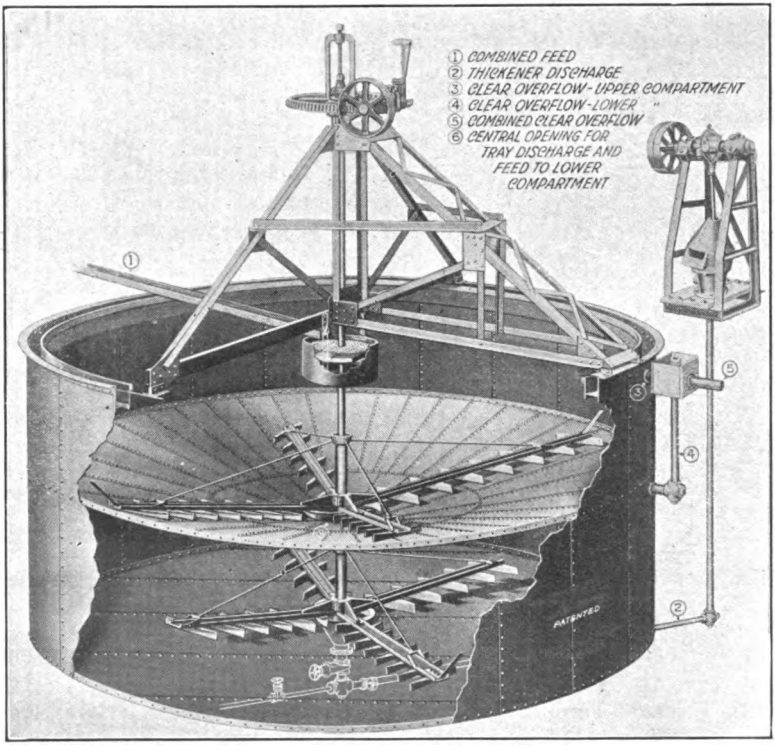

① COMBINED FEED
② THICKENER DISCHARGE
③ CLEAR OVERFLOW-UPPER COMPARTMENT
④ CLEAR OVERFLOW-LOWER "
⑤ COMBINED CLEAR OVERFLOW
⑥ CENTRAL OPENING FOR TRAY DISCHARGE AND FEED TO LOWER COMPARTMENT
PATENTED

FIG. 157.—Dorr tray thickener—open type.[80]

Conical Settlers (Fig. 158) are usually operated in parallel, as their capacity is small. Feed is delivered centrally, and the velocity of discharge of the thickened pulp may be lessened and fall required reduced by the use of a gooseneck discharge. Thickened pulp flow may be adjusted by means of a Bunsen valve operating on a piece of rubber tubing connecting any two convenient points in the gooseneck; or by means of the regulator

TABLE XC.—TYPICAL DORR THICKENER PRACTICE[80]

Mill	Sq. ft. settling area per ton solids thickened in 24 hr.	Remarks
San Rafael, Pachuca, Mex....	4.5	Tube-mill product, 75 per cent. —200 mesh. Discharge, 45.5 per cent. solids.
Liberty Bell, Telluride, Colo..	7.1	Tube-mill product, much light argillaceous slime. Discharge, 30 per cent. solids.
Batopilas, Chihuahua, Mex...	0.6 to 0.9	40-mesh product; 90 per cent. passing 100 mesh.
Zambona Dev. Co., Sonora, Mex.....	3.1	Tube-mill product. Discharge 40 per cent. solids.
Nova Scotia, Cobalt, Ont. ...	5.4	Tube-mill product, 88 per cent. —200 mesh, ore diabase. Discharge, 40 per cent. solids. Feed 6 : 1.
El Palmarito, Sinaloa, Mex...	4.5	Tube-mill product—pure quartzite, 97 per cent. —200 mesh. Feed 7:1. Discharge, 65–70 per cent. solids. (Continuous decantation.)
Amparo, Jalisco, Mex.....	4.9	Tube-mill product—siliceous—93.5 per cent. —200 mesh. Feed 24.5:1. Discharge, 23.5 per cent. solids—used to feed vanners.
Veta Colorado, Parral, Mex...	5.0	Tube-mill product—rather argillaceous—71 per cent. —200 mesh. Feed 11:1. Discharge, 33 per cent. solids for agitator. Have settled to 65 per cent. solids.

Mill	Sq. ft. settling area per ton solids thickened in 24 hr.	Remarks
Pennsylvania Steel Co., Lebanon, Pa.	2.9	Thickening ahead of vanner concentration; pulp in water. Feed 2.8 per cent. solid. Discharge, 10.6 per cent. solids. Overflow, 0.4 per cent. solids—extremely fine—which does not interfere with using water again in mill.
Nevada Consolidated Copper Co., Ely, Nev.		Each 17-ft. thickener supplies wash water for twenty Wilfley tables and occasionally a surplus for use as wash water on vanners, all without the necessity of pumping. One thickener (area 226 sq. ft.) has a greater capacity than twelve 8-ft. cones (combined area about 525 sq. ft.).
B. H. Proprietary, Block 10, Broken Hill, Australia		Dewatering slime from lead-zinc concentration mill—three 24′ × 10′ thickeners handle about 5000 tons of water per day, in which is contained some 50 tons of slime, giving a perfectly clear overflow and a thick slime discharge containing about 45 per cent. moisture.

shown in Fig. 159, which may be made of $\frac{5}{8}$-in. round iron. In the case where an exceptionally steady flow of ore is received by each cone the regulation may be done by means of a reducer or nozzle attached to the end of the gooseneck.

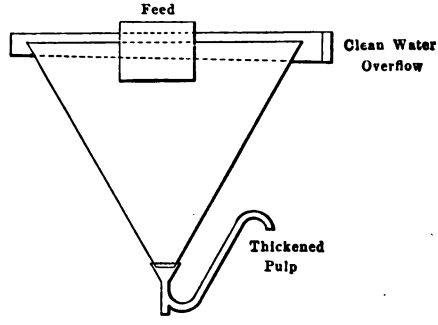


FIG. 158.—Cone thickener with gooseneck discharge.

Metal **Valves** are generally unsuitable for regulating the underflow from cones or spitzkasten. This is due to the fact that in throttling the discharge one or more sharp corners are formed

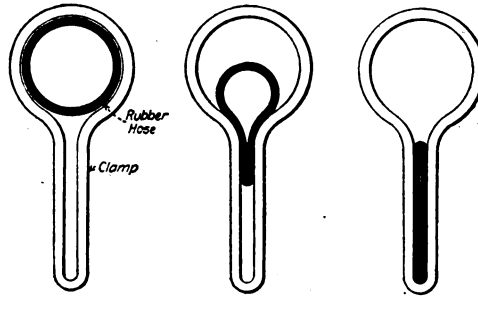


FIG. 159.—Hose discharge regulator.

which lead to an accumulation of solids and subsequent chokage of the cone. Stop-valves connected with the apices of cones should be of a type permitting no constriction in diameter

and should be left wide open during the operation of the cone.

Spitzkasten Settlers are usually operated in nests (Fig. 160). Angle of inclination of cone side depends on fall available and character of material to be thickened. Maximum possible slope is always advisable. With colloidal material there is a tendency

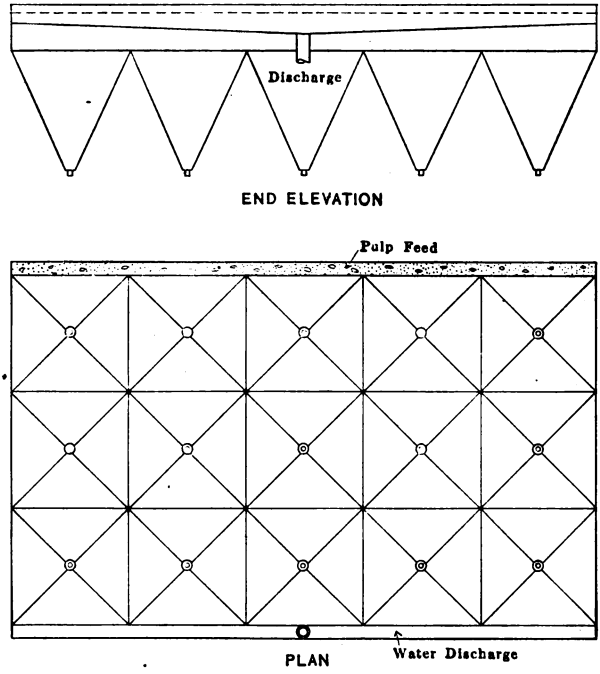


FIG. 160.—Spitzkasten settler.

for the slime to accumulate on the sides if the latter are insufficiently steep. This results in occasional chokage of the discharge. Goosenecks may be used in connection with spitzkasten settlers in the same way as with cones (*q.v.*).

Prism Spitzkasten Settlers are usually in series and are set transversely to the flow of pulp. An example of West Australian practice[7] shows the following dimensions:

Size, 111 ft. long by 42 ft. wide; number of prisms, 8; number of orifices in each prism, 30; distance between orifices, 3 ft. 8 in.; division plates between prisms, within 8 in. of surface of pulp; discharge, continuous from first row—intermittent from remainder; consistence of underflow, 1 to 1.

Details of design are shown in Fig. 161.

Gravitation Thickening may be carried out in any vat of suitable dimensions provided that the bottom slopes toward a point of discharge. If worked continuously there is an aggregation of solids in the bottom of the vat, and its capacity is rapidly reduced. By working intermittently and by sluicing out the accumulated slime by means of a high-pressure jet the apparatus may be operated at a fair efficiency.

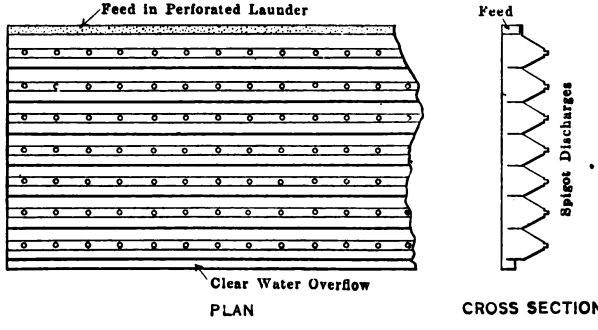


FIG. 161.—Prism spitzkasten settler.

The **Ayton** intermittent **Pulp Extractor,** for use with cones and pointed boxes, is a device consisting, in the main, of a valve the operation of which is controlled by a lever held in position by a spring and actuated by cam movement. The frequency and degree of opening of the valve are adjustable (see Fig. 162).

Dewatering is a term generally used in the case where it is possible to reduce the moisture percentage to a small amount, often as low as 5 per cent. This is effected directly by mechanical means, by pressure or vacuum involving filtration, and may be assisted by heat or compressed air.

Mechanical Dewatering of a **Sandy Product** may be effected in either of the mechanical-plus-gravitational classifiers previously described.

Dewatering by **Pressure** is effected by **Filter Presses** of the Dehne or Merrill types. In the former case the apparatus consists of a number of frames, usually about 4 ft. square and from 2 to 3 in. thick, separated by solid plates with channeled surfaces delivering to outlet ports at the bottom of each

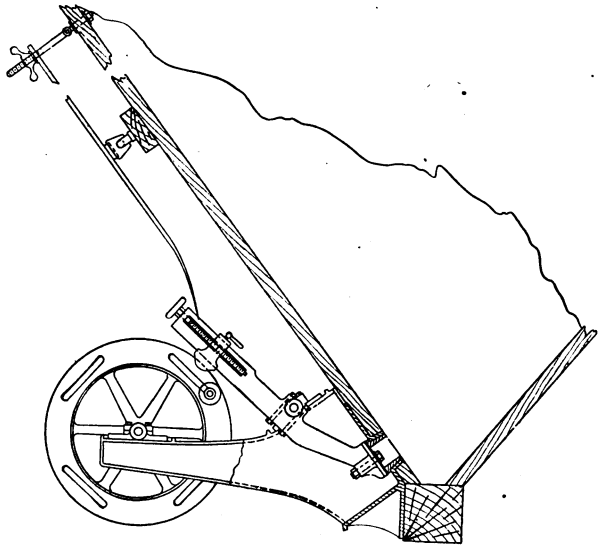


FIG. 162.—Ayton intermittent pulp extractor.[11]

plate. The latter are covered with suitable filter cloth, and the press is closed and the pressure joints are sealed by means of a hydraulic ram or pressure screws. Pulp is forced into the hollow frames by means of compressed air or pressure pump. When the frames are full and the cakes formed, the excess water may be removed by compressed air delivered through the filling port, or from alternate frames whose outlets have been temporarily closed. The press contents are discharged by releasing the pressure at the end, by drawing apart the frames and plates,

and by dumping the cakes of material into cars usually running on tracks below the press floor.

The **Merrill Filter Press** is only suitable as a dewaterer when subsequent treatment of the material can be effected by leaching in the press. The shape of the plate and frame is modified to allow for the introduction of a sluicing nozzle at the base of each frame, which rotates and automatically discharges the contents after treatment. Dewatering in this machine is only one phase

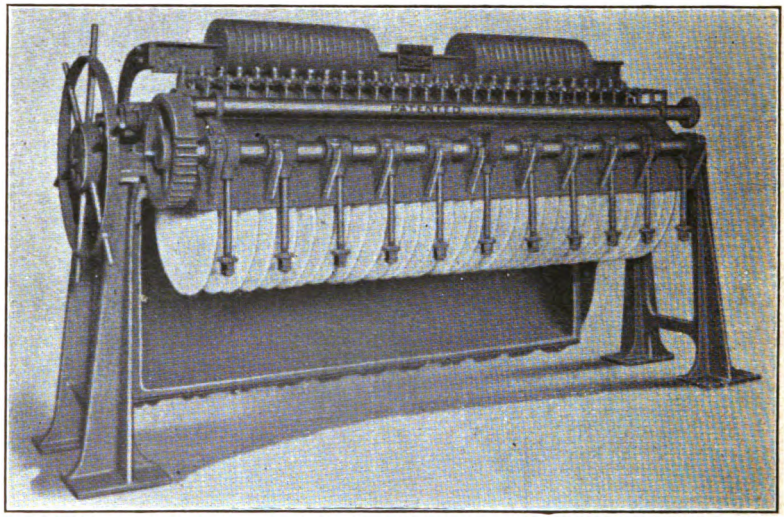


FIG. 163.—Sweetland filter press.[82]

of an operation which must necessarily terminate with as high or higher a moisture percentage in the discharged tailing as in the original pulp. At the Homestake, where this type of press is operated at a high efficiency for the treatment of low-grade slime, after amalgamation has recovered the bulk of the gold the presses are filled by gravity head pressure.

The **Sweetland Filter Press,** of intermittent operation, consists of a cylindrical filter body, divided on the horizontal center line into two halves which are hinged together (see Fig. 163).

A header is cast along the top half, through which holes are bored to receive the filter-leaf nipple outlets. The filter is locked by means of a series of swing bolts, which are tightened or loosened by means of an eccentric shaft. The filter leaf is a circular bag supported upon a wire-screen structure which prevents collapse of the bag and permits drainage of the clear liquor. In operation and after the displacement of the air the filter body is kept at a pulp pressure of about 40 lb. per sq. in. by means of gravity

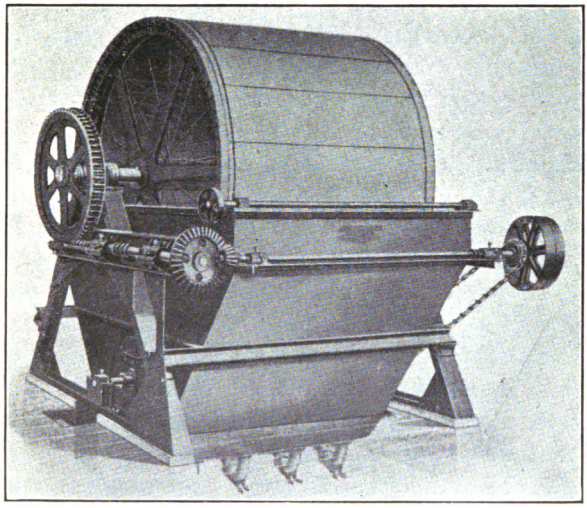


FIG. 164.—Revolving-drum type dewatering filter.[31]

pressure, pump, or montejus. The liquor passes out through the leaf and a cake is formed. The excess pulp in the shell may then be removed and the moisture percentage reduced by means of compressed air. Discharging is effected by opening the press, and by using a back current of air or steam to dislodge the cakes and free the pores of the filter medium.

Metallic Filter Cloth is an alloy in which nickel generally predominates, but may be made of copper, bronze, and other metals. This is drawn into fine wires, which are stranded and twisted.

The cloth is woven to give the desired texture and finally passed between rolls to flatten it and to produce the requisite degree of fineness.

Dewatering by Vacuum may be effected by means of a type of **Rotary Vacuum Filter** such as the **Oliver** or **Portland.** The principal requirements of such a machine when used for this purpose are: (*a*) automaticity and continuous action; (*b*) ample exposure to vacuum for drying without vibration or danger of cracking the cake; (*c*) provision for automatic discharge without necessity for air blast (which inevitably causes an increase in moisture percentage in discharged material by blowing back moisture already removed). With the rotary vacuum filter an air blow may be used to clear the pores of the canvas after the cake has been removed by scraper.

Fine Material to be dewatered in a **Rotary-Type Vacuum Filter** (see Fig. 164) is first thickened in a continuous machine, usually of the Dorr type. Details of the operation of Portland filters at the Butte & Superior mill on a flotation concentrate carrying 68.2 per cent. −200 mesh are given in Table XCI and show an average of under 8 per cent. moisture in the discharged product without steam-coil heating. With ordinary slimed ore it is practicable to reduce the moisture to from 10 to 20 per cent. with a machine of this type, according to the class of material being handled. In the case of a gold ore this permits efficient amalgamation and alkaline treatment, previous to cyanidation, without any subsequent accumulation of cyanide solution.

A combination **Dewatering Plant** for separate handling of coarse and fine material is shown in Fig. 165.

The **Shovel Wheel** for dewatering sand consists of a number of shovels attached to a wheel and which remove the sand from a trough. The shovels are usually perforated to facilitate drainage of the excess water and discharge of the sand. The **Cole Shovel Wheel** is shown in Fig. 166. This wheel is 24 in. in diameter and is fitted with 12 shovels of 1¼-in. scrap steel.

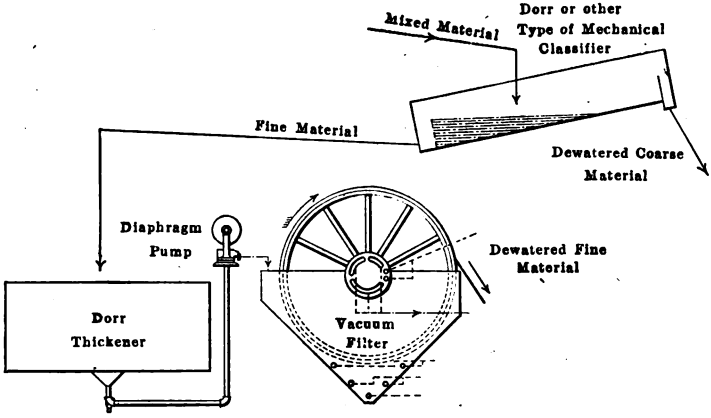


FIG. 165.—Combination dewatering plant.

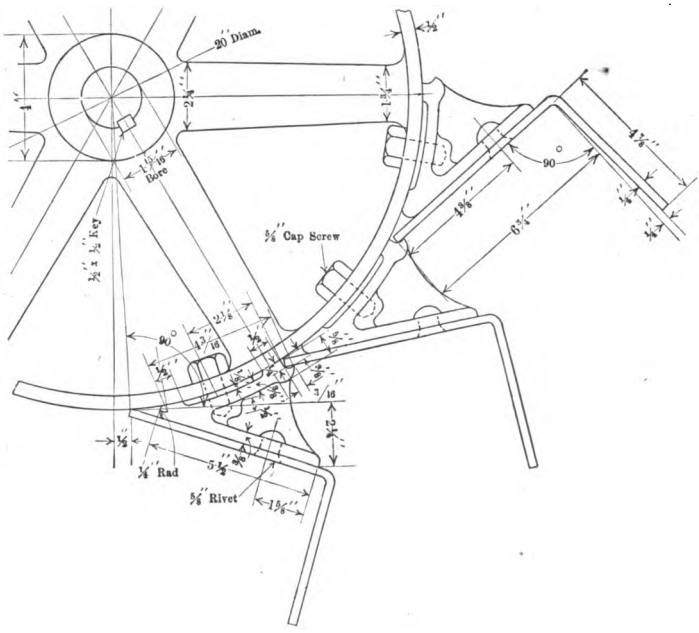


FIG. 166.—Cole shovel-wheel dewaterer,[83]

TABLE XCI.—PORTLAND FILTER DEWATERING TESTS AT THE BUTTE & SUPERIOR MILL

Test No.	Time		Vac., in.	Per cent., solid in feed	Thickness, cake, in.	Moisture, per cent.	Capacity per sq. ft. filter area per 24 hr., lb.
	Loading, min.	Drying, min.					
x2	½	5	22	61.91	½	9.72	972
3	½	6	22	61.91	⅜	8.41	672
x6	½	5	22	68.18	¾	6.89	1608
7	½	6	22	68.18	¾	5.93	1704
x8	½	6	22	68.18	⅝	6.36	1392
9	½	5	22	67.18	½	6.17	1128

NOTE.—Tests marked 'x' heated over steam coil while drying. Average capacity 14 ft. × 14 ft. Portland Filter with feed containing 44.6 per cent. solids, 194 tons per day; with feed containing 61.91 per cent. solids, 265 tons per day; with feed containing 67 to 68 per cent. solids, 403 tons per day. Average moisture in cake, without steam-coil heating, 7.87 to 7.28 per cent.

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